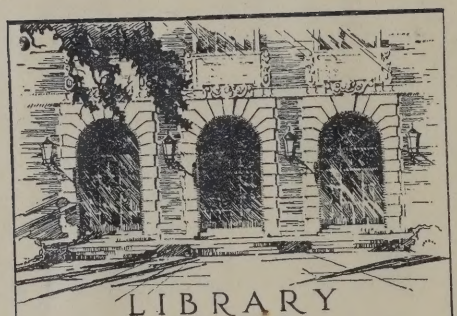

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AND OF THE

Mining Society of Nova Scotia

1952

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President

The Canadian Institute of Mining and Metallurgy

April 1952 to April 1953

Pilot Plant Investigations on Cold Water Separation of Bitumen from Alberta Bituminous Sands*

By L. E. DJINGHEUZIAN†

(*Ottawa General Meeting, January,* 1952)

(*Transactions, Volume LV,* 1952, pp. 1-14)

Introduction

FOR MANY YEARS the immense deposits of bituminous sands of northern Alberta have been attracting the attention of our governments, engineers and scientists (6). Undoubtedly these bituminous sands represent one of the greatest potential resources of liquid fuels in the world. However, their commercial development has been retarded by the complexities of the technological problem and by the economic factors involved in the processing of the sands. The technological problem has been attacked from several angles (2, 3, 1, 11), and cold water separation of bitumen is one of them.

To the ore dressing engineer, the cold water process is of utmost importance since, by the use of standard ore dressing machinery, it accomplishes most efficiently and economically the separation of bitumen from bituminous sands. The significance of using ore dressing equipment was especially enhanced by the fact that the scientific principles which underlie the design of standard machines were demonstrated in the cold water separation process with an extraordinary clarity. This greatly facilitated the analysing of limitations and advantages of different machines during the pilot plant investigations and helped to complete the investigations practically in the allotted time.

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†Metallurgical Engineer, Division of Mineral Dressing and Process Metallurgy, Mines Branch, Ottawa.

(6) For references, see end of paper.

Part I. — Cold Water Separation of Bitumen as an Ore Dressing Process

1. — *Pilot Plant Procedure and Results Obtained*

Pilot plant investigations on cold water separation of bitumen undertaken by the Bureau of Mines (now Mines Branch, Department of Mines and Technical Surveys), for which 150 tons of bituminous sands were obtained from the Horse River property of Abasand Oils, Limited, near McMurray, Alberta, were started on April 19th, 1949, and completed on May 19th, 1950 (Figure 1).

The purpose of these pilot plant investigations was to establish cold water separation of bitumen from Alberta bituminous sands as an ore dressing process, to work out the operational technique, and to establish the most economical flow-sheet possible.

To do this it was necessary to proceed in a systematic way, each week's runs being made under constant conditions so as to obtain reliable data. The performance of each machine had to be evaluated, basing these evaluations on the results of weekly runs. The characteristics of each procedure tried had to be determined with a view of deciding on a machine or a re-arrangement of the flow-sheet which might be an improvement on the previous one. Sampling procedure had to be most carefully standardized. In addition, the samples had to be assayed within 24 hours, for which purpose a special section was set up at the Division of Fuels for rapid analysis of the samples. A close check was kept on the controlled additions of the alkaline reagents and wetting agents, and effects of those additions studied.

In the pilot plant investigations, the following definitions have been used:

Diluent: Petroleum distillate which, added to the feed, would combine with liberated bitumen.

Oil: Combined bitumen and diluent

Crude Oil or Finished Product: No. 1 thickener overflow

Classifier Sands: Mineral matter eliminated by the classifier from the feed

Tails: No. 2 thickener underflow

Total Tails: Classifier sands + No. 2 thickener underflow

Classifier Extraction: Oil overflowed by the classifier and pumped into No. 1 thickener

Recovery: Oil overflowed by No. 1 thickener and delivered to the refinery

Actual Recovery: Bitumen input less total oil loss in terms of bitumen

Hence:

$$\%\ \text{sands eliminated by classifier} = \frac{\text{dry classifier sands}}{\text{mineral matter in feed}} \times 100$$

$$\%\ \text{classifier extraction} = \frac{\text{oil input} - \text{oil in classifier sands}}{\text{oil input}} \times 100$$

$$\%\ \text{indicated recovery} = \frac{\text{oil input} - \text{oil in total tails}}{\text{oil input}} \times 100$$

$$\%\ \text{actual recovery} = \frac{\text{bit. input} - \text{oil in total tails}^*}{\text{bitumen input}} \times 100$$

*In bitumen terms

The bituminous sands can be regarded as, essentially, a compacted mass of sand particles held together by bitumen which forms a film

Figure 1.—Shipment of 150 tons of bituminous sands in 500-lb. drums.

Figure 2.

1.—Soda ash feeder
2.—Feed belt
3.—Diluent feeder
4.—Agitator
5.—Pebble mill
6.—Rotary classifier

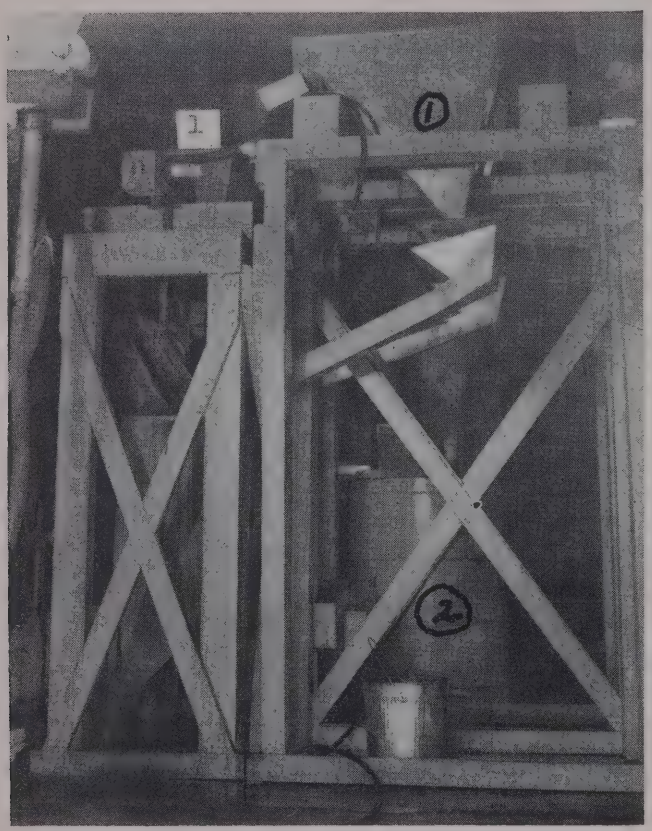

Figure 3.

1.—Cone classifier 2.—Settling tank

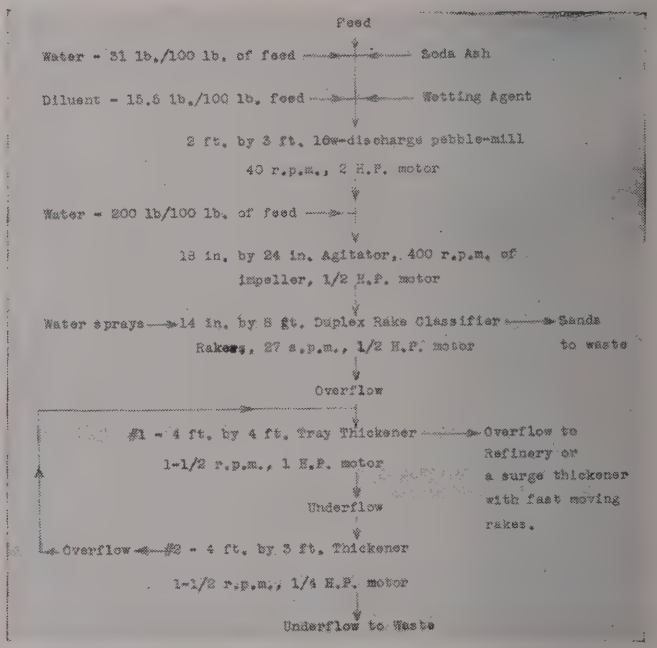
Feed
Water - 31 lb./100 lb. of feed
Soda Ash
Diluent - 15.6 lb./100 lb. feed
Wetting Agent
2 ft. by 3 ft. low-discharge pebble-mill
40 r.p.m., 2 H.P. motor
Water - 200 lb/100 lb. of feed
18 in. by 24 in. Agitator, 400 r.p.m. of impeller, 1/2 H.P. motor
Water sprays
14 in. by 8 ft. Duplex Rake Classifier
Rakes, 27 s.p.m., 1/2 H.P. motor
Sands to waste
Overflow
#1 - 4 ft. by 4 ft. Tray Thickener
1-1/2 r.p.m., 1 H.P. motor
Overflow to Refinery or a surge thickener with fast moving rakes.
Underflow
Overflow
#2 - 4 ft. by 3 ft. Thickener
1-1/2 r.p.m., 1/4 H.P. motor
Underflow to Waste

Figure 4.—Flow-sheet of pilot plant.

around each particle. When a light oil having the properties of kerosene is added to the bituminous sands at the same time that the bitumen film around the sand particles is ruptured or scoured off by some rubbing action, the bitumen dissolves in the light oil, forming a compound oil with a specific gravity of less than 1. When this process takes place in water, the heavier-than-water sand particles, liberated from oil, fall to the bottom of the vessel and the lighter oil rises to the surface. Thus fundamentally, the cold water process is a gravity concentration process in which the valuable constituent of the ore rises to the surface and the gangue sinks to the bottom.

In the preliminary stages of pilot plant investigations, the process consisted of the four following steps (Figures 2 and 3):

(1) Disintegration of bituminous sands by means of a low-discharge pebble-mill, kerosene in amount approximately equal to the bitumen content of the sands being fed into the pebble-mill together with the sands. Soda ash was also added to the feed.

(2) Separation of combined bitumen-kerosene, designated as oil, from sands in an agitator.

(3) Classification of agitator discharge, by means of a rotary classifier, into sands and oil slurry.

(4) Settling the oil slurry with cone classifiers and in tanks.

A product suitable for the heat-treatment plant (refinery), as designed by the Division of Fuels, was obtained.

The optimum temperature ranges were determined as lying between 73° and 81°F. Below 73°F., the separation became less efficient and incomplete. When the temperature approached 83°F., the sands discharged from the rotary classifier started becoming dirty and more free oil ran over with them, while higher temperatures resulted in a

TABLE I.— WEEKLY MILL RUNS DATA

Bituminous Sands Treated lb.	Rate of Feed lb./hr.	Soda Ash lb./ton	Wetting Agent lb./ton	Aver. Temp. °F.	pH of C.O'F*	Per Cent Bitumen in Tar Sands	Viscosity of Combined Oil Centipoises	Per Cent Sands Eliminated by Classifier	Per Cent Oil in Classifier Sands
Series of Weekly Runs With Kerosene									
6,550	238.1	1.46	Span 80 0.041	77.4	9.5	15.8	103.7	92.8	0.41
4,050	238.1	1.45	Rosin Amine 0.041	77.2	9.2	16.2	103.7	89.6	0.47
6,600	300.0	1.56	Span 80 0.031	78.3	9.3	14.9	103.7	88.3	0.38
3,100	215.0	1.51	Span 40 0.041	77.0	9.1	15.9	103.7	82.0	0.26
Series of Weekly Runs With Bitumount Diluent									
4,450	170.0	0.68	Span 80 0.062	79.0	8.6	15.4	336	87.7	0.26
5,700	215.8	0.89	Span 80 0.045	81.5	9.0	16.5	336	92.3	0.35

*Classifier overflow

still dirtier sands discharge. Thus for high bitumen recoveries, the temperature control of the classifier overflow was the key to efficient operation of the plant.

The other important feature of the separation was the work done by the low-discharge pebble-mill. It carried ½ lb. of pebbles to each lb. of bituminous sands fed per hour. The function of the pebbles was to disintegrate the sands and to scour the bitumen off the sand particles. A small amount of grinding took place, and the measure of this grind, as will be discussed later, became the measure of the completeness of scouring the bitumen off the sand.

In these preliminary runs, the indicated recoveries of oil were over 97 per cent, and samples of crude oil, taken with a dipper from a tank after overnight settlement, averaged 76.4 per cent oil, 1.9 per cent mineral matter, and 21.7 per cent water. The requirements of the refinery for the crude oil, as specified by the Division of Fuels, were "not less than 65 per cent oil and not more than 5 per cent mineral matter and 30 per cent water", and a crude oil suitable for treatment in the refinery was always obtained when settling was allowed. However, in a big plant, settling meant a very large tankage capacity, which, from the ore dressing point of view, did not appear to be attractive. Hence it was decided that, while keeping the recoveries as high as possible, all efforts should be directed toward obtaining a crude oil of sufficiently high grade without the necessity of settling.

It turned out that this was a wise decision. As the investigations progressed, it was found that settling caused excessive oil losses through the formation of an oil emulsion termed 'sludge' which settled at the bottom of the tanks. When attempts were made to adapt the ore dressing technique, already worked out, to the recovery of oil from the sludge they proved ineffective. In these attempts, both a scavenging procedure and adding the sludge to the bituminous sands were tried.

In the scavenging runs, one part of sludge was mixed with two parts of clean classifier sands, and this mixture was used as the mill feed. The runs were made with and without additions of kerosene, but in either case scarcely any oil was recovered from the sludge. In the second trial, the mill feed consisted of one part of sludge and two parts of bituminous sands. However, the addition of the sludge to the bituminous sands produced disastrous results — only about two-thirds of the diluent input and bitumen in the sands was recovered, while no oil was recovered from the sludge.

Thus it appeared that the sludge was a stable emulsion, the addition of which to the mill circuit had a very pronounced detrimental effect on bitumen recoveries. No further experiments with the sludge were made because, even before they were started, the principal object of the investigations had been attained, namely, using kerosene as a diluent, a crude oil of a grade sufficiently high to be suitable as a direct feed to refinery was produced in a continuous process without settling.

This goal of obtaining a crude oil of refinery grade, together with high recoveries, was finally achieved when diaphragm suction pumps replaced vertical centrifugal pumps, and suitable wetting agents were used together with soda ash.

Figure 4 shows the flow-sheet with which the pilot mill runs were successively completed:

Table I gives the results obtained with the last six weekly series of runs. Of these, the first four were made to test the effectiveness of different wetting agents, using kerosene as a diluent, and the last two when using as a diluent a heavier-than-kerosene petroleum distillate, obtained from the experimental plant operated by the Alberta Government at Bitumount, Alberta.

Several observations which sum up the results obtained from numerous daily runs can be made from a study of this table:

(1) In every series of runs, with high overall recoveries maintained, a thickener overflow was obtained which could be fed directly into the refinery.

(2) The optimum amounts of soda ash required per ton of tar sands are governed by the *pH* of the classifier overflow. Whenever this *pH* dropped below 9.0 the recoveries started to fall off.

(3) The beneficial effects of all three wetting agents on the treatment of tar sands were firmly established. Indications were that

TABLE I.— WEEKLY MILL RUNS DATA

Per Cent Extraction of Oil by Classifier	Per Cent Oil in No.2 Thick. U.F.— Tails	Tails to Waste lb.	Per Cent Indicated Recovery	Per cent Actual Recovery	Loss of Bitumen lb./ton	Loss of Diluent lb./ton	No. 1 Thickener Overflow Per Cent		
							Oil	M.M.	Water
Series of Weekly Runs With Kerosene									
98.6	0.12	14,960	97.8	95.6	7.2	6.8	71.3	2.1	26.6
98.6	0.17	11,000	97.1	94.2	9.4	8.9	71.5	1.3	27.2
98.8	0.12	18,040	97.7	95.4	6.9	7.4	75.6	2.2	22.2
99.3	0.10	10,120	98.2	96.4	5.7	5.6	75.6	1.7	22.7
Series of Weekly Runs With Bitumount Diluent									
99.2	0.18	20,240	96.5	93.0	10.6	10.5	67.5	3.0	29.5
99.0	0.12	22,880	97.5	95.0	8.1	8.5	70.9	2.7	26.4

Span 40 gave the best results, followed by *Span* 80 and *Rosin Amine D Acetate*. For a large-scale operation, these three wetting agents should be tried over longer periods so as to find out which would give the best results. The cost of *Span* 40 and *Span* 80 is the same and comes to about $0.33 per lb. The cost of *Rosin Amine D Acetate* is $0.22 per lb.

The amounts of these wetting agents used were based on the amounts of water added to the pebble-mill feed. The laboratory experiments indicated that the optimum amount of wetting agent required was 2.1 grams per 67.7 lb. of water which was fed, together with 225 lb. of bituminous sands, into the pebble-mill per hour, or 0.041 lb. of wetting agent per ton of tar sands. When the feed rate was increased to 300 lb. per hour, the amount of *Span* 80, for experimental purposes, was not increased proportionately to water added to the pebble-mill. This reduced the amount of wetting agent to 0.031 lb. per ton, which did not affect the overall recovery and appeared to improve the grade of the thickener overflow.

(4) The indicated recoveries of oil by the use of the Bitumount diluent were practically the same as with kerosene, and, more important, the grade of the thickener overflow was suitable as a refinery feed so that settling was unnecessary.

Since the diluent to be used in a large commercial operation must be a derivative from the bitumen itself, it was essential to ascertain whether the ore dressing technique, already worked out with kerosene as a diluent, would apply also when using heavier oils directly derived from bitumen. Six barrels of an oil derivative from Alberta bituminous sands were obtained from the Bitumount plant in Alberta. This oil, with a specific gravity of 0.88, when mixed in equal weight proportions with the bitumen, would produce an oil of a calculated specific gravity of 0.936, whereas the specific gravity of the kerosene-bitumen mixture was 0.89.

(5) During the fifth run of the last series, the average temperature of the classifier overflow was 86.9°F. and the oil content of the classifier sands 0.9 per cent, as against 80°F. and 0.2 per cent in all the previous runs of this series. This brought the average temperature of the series up and lowered the classifier extraction. It was another indication that whenever the temperature of the classifier overflow exceeded 83°F., the classifier sands would become dirty and carry more oil.

2. — *Analysis of the Performance of Principal Machines Used in the Flow-Sheet*

(*a*) 2 *ft. by* 3 *ft. Low-discharge Pebble-Mill*

This mill was a Patterson tube-mill, powered with a 2 h.p. motor and running at 40 r.p.m. The inside dimensions of the mill, when newly relined with 2-inch-thick Porox blocks, were: diameter, 20 inches; length, 30 inches. When newly relined, the mill was operated at 67.4 per cent of the critical speed (Figure 5).

The discharge-end liners consisted of six Porox sections, each with six round holes, through which the oil slurry passed to be directed by lifters into the discharge trunnion. These holes were tapered, their diameters on the outside and inside being 1⅛ in. and 1 in., respectively.

The mill was started with a 100 lb. charge of Danish pebbles as the disintegrating and partly grinding media. During the first few runs the pebble charge was stabilized, using kerosene as a diluent, at ½ lb. of pebbles per lb. of bituminous sands fed per hour. After run No. 30, the Danish pebbles were replaced by Porox balls. However, the consumption of Porox balls was found to be so high that, after 57 runs, the use of Danish pebbles was resumed and continued until the pilot plant runs were completed.

The bituminous sands were fed into the tube-mill automatically by means of a screw conveyor, fitted to the feed-end of the mill. This screw conveyor was operated by ½ h.p. gear motor. It must be said here that, after completion of the runs, in which 69 tons of bituminous sands were treated, the screw conveyor showed scarcely any sign of wear.

The laboratory tests indicated that the time of grind required for obtaining the highest recoveries of bitumen was around 20 minutes. However, it was decided to base the study of the tube-mill performance not on the time of residence but on the actual grind done by the mill, since it was thought that the meas-

ure of grind would be equivalent to the measure of loosening and scouring the bitumen off the sands. Accordingly, the Fuel Research Laboratories were asked to make screen analyses on weekly composite samples of the tube-mill feed and on daily samples of the tube-mill discharge.

From a study of weekly and daily screen analyses of mill feed and tube-mill discharge, respectively, it was concluded that, with all the wetting agents used and the variable clay contents, the optimum grind for the best recovery of oil prepared to be equivalent to a reduction of mineral matter in the sands from 5-7 per cent minus 200 mesh to 11-15 per cent minus 200 mesh. This conclusion was confirmed by runs in which insufficient grinding failed to scour the whole of the bitumen off the sands, and by others in which over-grinding, by producing more fines, was detrimental to overall oil recoveries. Close control of grinding was shown to be of primary importance for the efficient treatment of the sands.

Table II shows the extreme variations obtained in weekly screen analyses of tube-mill feed and of tube-mill discharge, with corresponding grinds, for runs of series 2 and series 4. The pebble charge was the same in both series of runs.

After stabilizing the pebble charge at ½ lb. pebbles per lb. of bituminous sands fed per hour, the rate of feed was gradually increased from 150 lb. to 300 lb. per hour, ½ lb. pebbles being added to the tube-mill for every 1 lb. increase in the rate of feed.

By raising the rate of feed in this manner, no difficulties were encountered in grinding. However, when kerosene as a diluent was replaced by the Bitumount diluent, the tube-mill, with the same pebble charge, quit grinding and became overloaded. This situation was remedied by increasing the pebble charge to 0.6 lb. pebbles per lb. of feed per hour.

Table III shows the screen analyses of tube-mill feed and discharge obtained in series 4 and series 6, together with some pertinent data.

Studying this table, the conclusion may be drawn that practically optimum grinds were obtained in both series.

Power

The tube-mill motor was watched and was found to be always cool, no signs of overloading of the motor being observed at any time. A determination of power was made when the pebble charge was 130 lb. and feed rate 225 lb. of bituminous sands per hour. The reading was 1½ amperes, or approximately 1.8 h.p.

In their catalogue, the manufacturers of Patterson mills give the data on power required by different sized Porox-lined pebble-mills. From these data we find the following: a 2 ft. by 3 ft. mill with a 50 per cent pebble charge will draw approximately 1½ h.p., whereas an 8 ft. by 12 ft. mill will draw 60 h.p., or 1 h.p. per 400 lb. of pebble charge. Since the viscosity of oil, which causes the formation of a viscous mass inside the mill, must be taken into consideration, 1 h.p. per 300 lb. of pebble charge appears to be a safe figure in considering the horsepower required for a large commercial mill with high discharge. This horsepower should be increased by 30 per cent if a low-discharge mill is used.

TABLE II.—SCREEN ANALYSES OF TUBE-MILL FEED AND DISCHARGE

MESH	TUBE-MILL FEED		TUBE-MILL DISCHARGE	
	% WEIGHT	CUM. % WT.	% WEIGHT	CUM. % WT.
SERIES 2				
+60	1.4	1.4	0.5	0.5
+80	7.2	8.6	2.5	3.0
+100	14.5	23.1	11.9	14.9
+150	58.4	81.5	49.2	64.1
+200	13.9	95.4	22.3	86.4
−200	4.6	4.6	13.6	13.6
	100.0		100.0	
SERIES 4				
+ 60	1.2	1.2	1.0	1.0
+ 80	9.2	10.4	5.5	6.5
+100	19.5	29.9	14.7	21.2
+150	51.2	81.1	47.7	68.9
+200	12.1	93.2	17.0	85.9
−200	6.8	6.8	14.1	14.1
	100.0		100.0	

TABLE III.—SCREEN ANALYSES AND OPERATING DATA

	SERIES 4		SERIES 6	
	FEED, %	DISCH. %	FEED, %	DISCH. %
+100	29.9	21.2	25.4	21.0
+150	51.2	47.7	57.2	46.4
+200	12.1	17.0	12.4	18.4
−200	6.8	14.1	5.0	14.2
Diluent used	Kerosene		Bitumount	
Wetting agent used	Span 40		Span 80	
Viscosity of oil, centipoises	103.7		336	
Lb. bituminous sands per hour	215.0		215.8	
Weight of pebble charge, lb.	110		135	
Lb. pebbles per lb. of feed/hr.	0.53		0.63	
% overall recovery of oil	98.2		97.5	

Grinding Media

Since pebble consumption is accountable for an important part of the cost of disintegration and grinding of the sands, a close check was kept on this item. Table IV gives data on the consumption of Porox balls and Danish pebbles.

It must be noted here that, before the final weighing of Danish pebbles, all flattened pebbles were discarded, thus giving a high figure for pebble consumption. In large-scale commercial operations, a lower pebble consumption should be expected as the Danish pebbles will clear themselves out.

At the same time, since the cost of Danish pebbles appears to be quite a large item, other possible supplies of pebbles should be considered. Fortunately, suitable pebbles are found in Alberta, not very far from the site of bituminous sands deposits. The cost of these pebbles, after screening, would be

somewhere between $10 and $20 per ton delivered at the plant, as against $80 per Danish pebbles. These Alberta pebbles were not tested in our pilot plant; nevertheless, since very little grinding is done in the mill, they could be used for the disintegration of tar sands. Even assuming that their consumption will be higher than that of Danish pebbles, the cost differential will be great enough to make them much more economical than Danish pebbles.

Mill Lining

The shell lining of the mill, as mentioned before, consisted of 2-inch Porox blocks. After 64 tons of bituminous sands were treated it was necessary to re-line the shell of the mill with new Porox blocks. In one place, the shell lining was worn down to 3/8 in. thickness, and in this place a piece of lining was broken off, leaving part of the shell of the mill unprotected.

Naturally, in a large commercial mill, Porox lining consumption will be considerably lower than in our small Patterson mill, but since its cost is $0.24 per lb., its use in the treatment of the bituminous sands appears to be prohibitive. In any future operations, Silex lining or Ni-Hard liners should be used. The cost of Silex (Buhrstone) lining is 6½ cents per lb. and, according to manufacturers, it has generally four times the life of Porox. Based on these figures, Silex works out to be almost 15 times cheaper than Porox.

TABLE IV.— CONSUMPTION OF GRINDING MEDIA

	POROX BALLS	DANISH PEBBLES
Total tons of bituminous sands treated	28.725 tons	20.77 tons
Total consumption of balls or pebbles	89.50 lb.	26.75 lb.
Consumption of balls or pebbles per ton of sands	3.12 lb.	1.29 lb.
Size of new balls or pebbles added	2 in.	2½ to 3 in.
Cost of balls or pebbles per lb.	$0.22	$0.04
Cost of balls or pebbles per ton of bituminous sands	$0.686	$0.052

(b) 18 in. by 24 in. Denver Laboratory Agitator

This agitator was equipped with a ½ h.p. motor, and the speed of its impeller was about 400 r.p.m. Since the oil slurry was discharged from the lower opening of the agitator, or 12 inches below the top, the volumetric capacity used was just over 1.5 cu. ft., the total capacity of the agitator, specified by the manufacturers, being 3.28 cu. ft. (Figure 5).

If to 300 lb. of bituminous sands per hour about 45 to 50 lb. of diluent and 770 lb. of water are added, then the volume of oil slurry would work out to be 15.7 cu. ft. per hour, or the time of contact of the sands in the agitator would be 5.7 minutes. This would give 2,280 revolutions of the impeller per contact, at which rate the agitator did an excellent job. However, the laboratory tests indicated that 500 revolutions per contact were sufficient for effective separation of the oil from the mineral matter.

Thus, it would appear that the time of contact required is equal to 500/400, or 1.25, minutes, and that, based on the laboratory tests, the indicated capacity of the agitator, when the oil slurry is discharged from the lower opening, is around 1,368 lb. per hour or, say, 16 tons per 24 hours.

Only large-scale tests could confirm this. Therefore, in the design of a 500-ton plant, calculations of the size of the agitator should, as a safety measure, be based on the performance of the 18 in. by 24 in. agitator. In other words, if 400 r.p.m. is the speed of the impeller, a 7 ft. dia. by 7 ft. high tank should be used. Naturally, by increasing the size of tank, the speed of the impeller would be reduced proportionately, which would cut down on power requirements.

14 in. by 8 ft. Dorr Duplex Rake Classifier

To effect the primary separation of oil from the sands, a 14 in. by 8 ft. Dorr Duplex rake classifier was

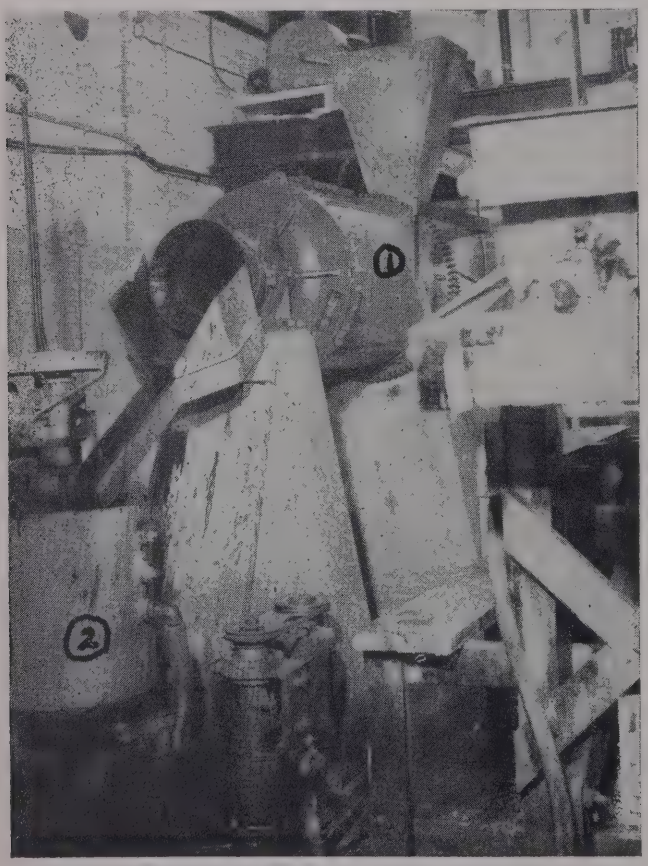

Figure 5.

1.—Low discharge pebble mill

2.—Denver laboratory agitator

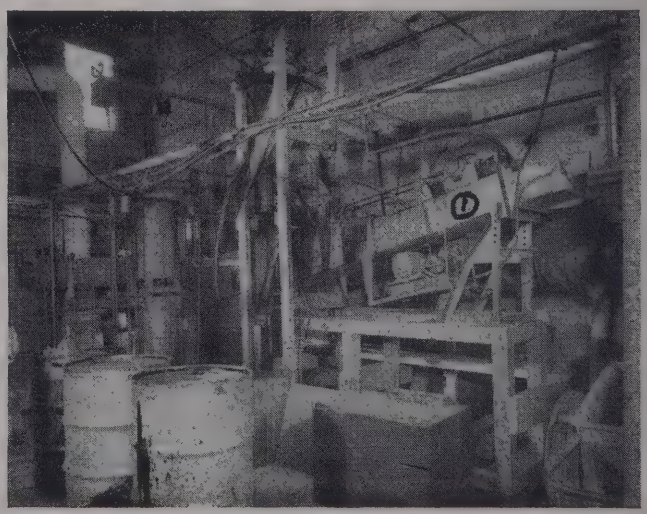

Figure 6.

1.—Duplex rake classifier

used, into which was pumped the oil slurry from the 18 in. by 24 in. agitator. This classifier was equipped with a ½ h.p. motor, the rakes making 27 s.p.m. It did an excellent job on feeds ranging from 225 to 300 lb. of bituminous sands per hour (Figure 6).

In cold water separation of bitumen, the function of a classifier is twofold: not only should it eliminate the maximum amount of mineral matter from oil slurry, discharging the mineral matter in the form of sands, but, in addition, these sands should contain negligible amounts of oil, so that they would not require further treatment and could be sent directly to waste. Thus, actually, a classifier becomes an extraction machine.

Figures showing the classifier performance (per cent oil entrained in the sands, per cent elimination of sands from mineral matter in the bituminous sands, and per cent extractions of oil by the classifier) have already been given in Table I and need not be repeated here. Those figures show that almost 90 per cent of the mineral matter in the bituminous sands is eliminated by the classifier as waste, with only about 1 per cent loss of total oil input (Figure 7). This makes thickeners, which can be operated at low cost, eminently suitable for efficient recoveries from classifier overflow of oil of a sufficiently high grade to be conveniently treated in the refinery.

Observations during the series of runs in which the feed rate was 300 lb. per hour indicated that the full capacity of the classifier might have been reached. Hence, it appeared that in calculations for a classifier with a capacity of 500 tons of bituminous sands per 24 hours, a capacity of 300 lb. per hour, or 3.6 tons per 24 hours, should be assigned to a 14 in. by 8 ft. classifier. This was subsequently confirmed when mill runs with a feed rate of 450 lb. per hour were made.

No. 1 and No. 2 Thickeners (Figure 8)

No. 1 thickener was a 4 ft. dia. by 4 ft. deep Dorr tray machine, powered with a 1 h.p. motor and equipped with a simplex diaphragm pump with a separate ¾ h.p. motor. The rakes of the thickener made 1½ revolutions per minute.

The overflow of this thickener (see Table I) contained over 70 per cent oil and was a product suitable for refinery treatment. The daily underflows contained anywhere from 0.2 to 2.3 per cent oil, the averages for each series of runs varying from 0.7 to 1.2 per cent oil. Hence the underflow could not be sent to waste and had to be subjected to a further thickening operation.

Table V.—Thickener Recoveries of Oil

Amount of oil on No. 1 thickener feed	1,893.9 lb.
Weight of No. 1 thickener underflow (slightly underestimated)	23,042 lb.
% oil in No. 1 thickener underflow	0.7 %
Total oil in No. 1 thickener underflow	161.3 lb.
% recovery of oil in No. 1 thickener	91.5 %
Weight of No. 2 thickener underflow (tails)	22,880 lb.
% of oil in No. 2 thickener underflow	0.12 %
Oil lost in tails	27.3 lb.
Oil recovered by No. 2 thickener from No. 1 thickener underflow	134.0 lb.
% oil recovered by No. 2 thickener from No. 1 thickener underflow	83.1 %
% additional overall recovery of oil by No. 2 thickener	7.1 %
% indicated overall recovery of oil by both thickeners from classifier overflow	98.6 %
Total tar sands treated	2.85 tons
Oil, bitumen plus diluent, recovered per ton by No. 2 thickener	47.0 lb.

Figure 7.—Rake classifier discharging clean sands.

Though No. 1 thickener did excellent work in producing a suitable grade of crude oil and gave high indicated recoveries, nevertheless, being a tray thickener, it was definitely unsuitable for the job.

When oil which sank with mineral matter to the bottom of the thickener rose on being liberated by the action of the rakes it was trapped under the tray, where it accumulated. The accumulated oil was of low grade and had a high content of mineral matter, which was due to the fact that some of the descending mineral matter was caught and retained by the liberated oil ascending under the tray. It was concluded, therefore, that for all future operations a thickener without trays should be used, and also that it should be deeper, as with a deeper thickener there is a probability that the second stage thickening would not be required.

In the pilot plant operations, a deeper than 4 ft. thickener not being available for trial, it became necessary to use another shallow thickener in series with the first one. This second thickener was a 4 ft. dia. by 3 ft. deep Denver spiral rake machine, powered with ¼ h.p. motor. The underflow was discharged by gravity and sent to waste, and the overflow was returned by gravity to the first thickener.

Table V shows the results obtained with No. 2 thickener in the last series of runs, when the Bitumount diluent was used.

It can be seen from the figures in Table V that No. 2 thickener was performing a very useful function. It recovered 83.1 per cent of the oil from No. 1 thickener underflow, thus raising the indicated overall recovery by 7.1 per cent and giving an additional recovery of 47 lb. of combined oil per ton of bituminous sands.

Preliminary studies indicated that if oil recovered in the process is to be used as fuel for the generation of steam power to treat bituminous sands, then about 6.4 lb. of recovered oil would be required as fuel for the treatment of one ton of bituminous sands. This alone makes the use of the second thickener necessary. The economic soundness of having the second stage of thickening is emphasized by the fact that 40 odd lb. of oil per ton of bituminous sands was recovered, in addition to oil that would be required as fuel for the generation of power. On the other hand, since the combined depth of the two thickeners was 7 feet, it is possible that, in a commercial operation using 8 ft. deep thickeners, one stage of thickening might be sufficient. However, this can be ascertained only from actual operation, since with two thickeners in series we have the combined action of two sets of rakes at the bottom of thickeners, and having two sets of rakes might be a requisite for obtaining high overall recoveries.

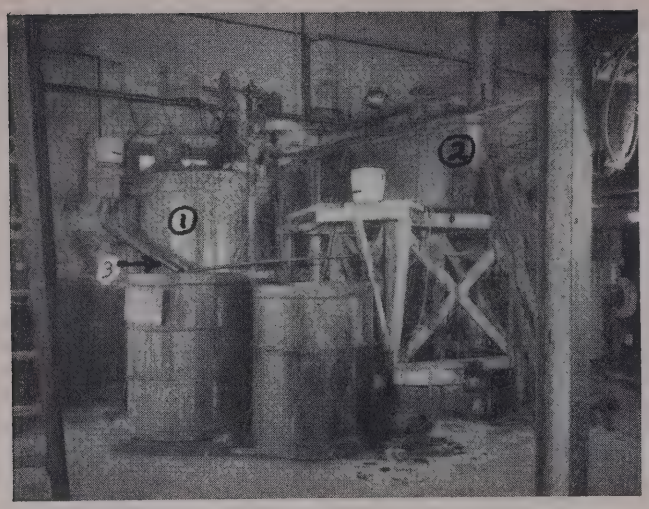

Figure 8.

1.—No. 1 thickener 2.—No. 2 thickener

3.—Crude oil discharge

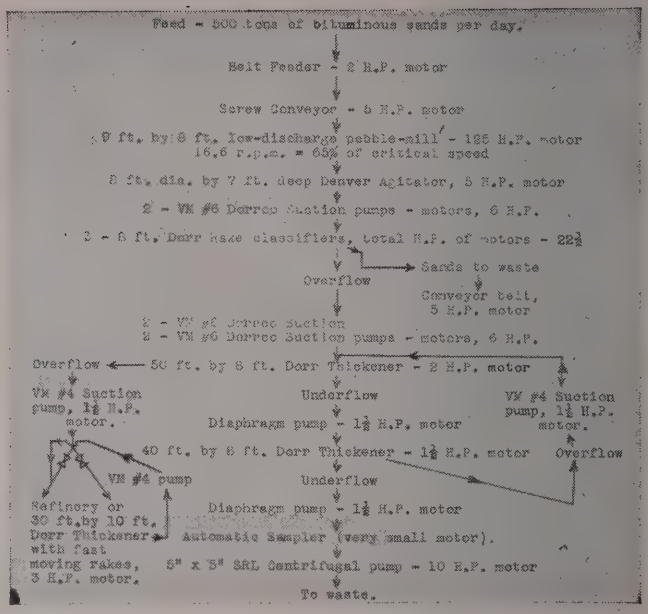
Feed - 500 tons of bituminous sands per day.
Belt Feeder - 2 H.P. motor
Screw Conveyor - 5 H.P. motor
9 ft. by 8 ft. low-discharge pebble-mill - 125 H.P. motor
15.6 r.p.m. = 65% of critical speed
8 ft. dia. by 7 ft. deep Denver Agitator, 5 H.P. motor
2 - VM #6 Dorrco Suction pumps - motors, 6 H.P.
2 - 6 ft. Dorr Rake classifiers, total H.P. of motors - 22½
Sands to waste
Overflow
Conveyor belt, 5 H.P. motor
2 - VM #6 Dorrco Suction
2 - VM #6 Dorrco Suction pumps - motors, 6 H.P.
Overflow
50 ft. by 8 ft. Dorr Thickener - 2 H.P. motor
VM #4 Suction pump, 1½ H.P. motor.
Underflow
VM #4 Suction pump, 1½ H.P. motor.
Diaphragm pump - 1½ H.P. motor
40 ft. by 8 ft. Dorr Thickener - 1½ H.P. motor
Overflow
VM #4 pump
Underflow
Refinery or 30 ft. by 10 ft. Dorr Thickener with fast moving rakes, 3 H.P. motor.
Diaphragm pump - 1½ H.P. motor
Automatic Sampler (very small motor).
5" x 3" SRL Centrifugal pump - 10 H.P. motor
To waste.

Figure 9.—Flow-sheet of proposed 500-ton/24 hr. pilot plant.

Total installed horsepower, 201 h.p.

3. — Mill Runs to Determine the Capacity of the Cold Water Separation Plant

Two mill runs were made with a view to determining definitely the capacity of the pilot plant.

Since 270 lb. of pebbles would occupy 50 per cent of the tube-mill volume, the feed rate used was 450 lb. of bituminous sands per hour.

Rate of Feed

Span 40	0.041 lb./t
Soda ash	1.5 lb./t
Kerosene	13% of wt. of feed
Water	32% of wt. of feed
Av. t° during runs	79.7°C.
Av. *pH* class. o'flow	9.2
Ratio bitumen: kerosene	54.3:45.7

Thus a 50:50 ratio was not maintained. This was due to the fact that the maximum amount of kerosene that could be fed with the feeder on hand was 58.4 lb./hour.

Table VI summarizes the results obtained.

Table VI.— Results Obtained with Feed Rate of 450 Lb./Hr.

Total bituminous sands treated	4,050 lb.
Feed rate per hour	450 lb.
% bitumen in the sands	15.4 %
Total estimated amount of bitumen in the sands	624.1 lb.
Kerosene added	525.6 lb.
Total estimated oil input	1,149.7 lb.
Weight of Dorr classifier wet sands	3,927 lb.
% oil entrained in classifier sands	0.59 %
Total oil entrained in classifier sands	23.4 lb.
Oil extracted by classifier	1,126.3 lb.
% extraction of oil	98.0 %
% dry sands eliminated by Dorr classifier from mill feed	84.8 %
Weight of No. 2 thickener underflow (tails)	7,920 lb.
% oil in tails	0.6 %
Oil lost in tails	47.5 lb.
Overall oil losses	70.9 lb.
Indicated oil recovery	1,078.8 lb.
% indicated oil recovery	93.8 %
% actual recovery of bitumen	87.6 %
Bitumen loss per ton of bituminous sands	19.0 lb.
Kerosene loss per ton of bituminous sands	16.0 lb.
No. 1 thickener overflow: % Oil	71.1 %
% M.M.	3.5 %
% H_2O	25.4 %

Operational Summary

Performance of the Tube-Mill

Rate of feed: 450 lb./hour, or 5.4 tons/24 hours

Screen Analysis

Table VIII shows the screen analyses of the mineral matter in the tube-mill feed and discharge. They indicate that optimum conditions were obtained for the treatment of tube-mill discharge in the agitator and classifier.

Table VII.— Screen Analysis at Feed Rate of 450 Lb./Hr.

Mesh	Weights, Per Cent	
	Feed	Discharge
+20	0.6	0.2
−20+40	0.5	0.4
−40+60	0.5	0.7
−60+80	4.6	4.5
−80+100	16.8	12.0
−100+150	56.3	52.0
−150+200	14.8	15.6
−200	5.9	14.6
	100.0	100.0

Power and Capacity

Amperes drawn, 1.8, or h.p. consumed, 2.2.

The h.p. consumed when the mill carried 130 lb. of pebbles and treated 225 lb. of bituminous sands per hour was 1.8. By increasing the rate of feed by 225 lb./hour, or 2.7 tons/24 hours, the horsepower consumption was increased by 0.4. In other words, every 6.75 tons increase in feed per 24 hours will mean an increase of 1 h.p. in power consumption. Hence, to treat 500 tons will require 74 net h.p., or, since the net horsepower required is approximately 75 per cent of the gross power input, the gross power required will be around 100 h.p.

As regards the capacity of our 2 ft. by 3 ft. tube-mill used as a low-discharge pebble-mill, it was definitely established as 5.4 tons of bituminous sands per 24 hours. From this it can be calculated that the minimum capacity of an 8 ft. by 12 ft. or a 9 ft. by 8 ft. low-discharge pebble-mill is 500 tons per 24 ..

Performance of the Dorr Classifier

Practically from the very start it became evident that the classifier could not efficiently handle the increase in feed. Hardly any sprays could be used, and the discharged sands became dirty. The classifier extractions dropped by more than one per cent.

However, in a commercial operation it might be considered feasible to put higher tonnages through classifiers at lowered extractions, since the decrease in extractions will be offset by the decrease in capital and operating costs.

Performance of the Thickeners

Both thickeners failed to handle the increase in the tonnage. Table VIII shows the results obtained.

Studying the figures in Table VIII, it becomes obvious that a thickener in the cold water process, as in cyanidation, can do just so much and no more It can be seen that No. 2 thickener, being of the same diameter as No. 1 but shallower by 1 foot, recovered 462.1 lb. of oil as against 616.7 lb. by No. 1 thickener.

Conclusion

The results obtained in the above runs, in which a feed rate of 450 lb./hour was used, appear to establish the capacities of ore dressing machinery for the cold water process, and undoubtedly serve as a check on the design of a 500-ton flow-sheet.

An important feature of the performance of the thickeners was that, even at an increased rate of feed, the overflow of No. 1 thickener still constituted a crude oil of sufficiently high grade to be fed directly into the refinery.

TABLE VIII.—THICKENER RECOVERIES WITH FEED RATE OF 450 LB./HR.

Amount of oil in No. 1 thickener feed	1,126.3 lb.
% oil in No. 1 thickener feed	12.0 %
Weight of No. 1 thickener underflow (slightly underestimated)	8,355.0 lb.
% oil in No. 1 thickener underflow	6.1 %
Total oil in No. 1 thickener underflow	509.6 lb.
% recovery of oil in No. 1 thickener	54.8 %
Weight of No. 2 thickener underflow (tails)	7,920.0 lb.
% oil in No. 2 thickener underflow	0.6 %
Oil lost in tails	47.5 lb.
Oil recovered by No. 2 thickener from No. 1 thickener underflow	462.1 lb.
% oil recovered by No. 2 thickener from No. 1 thickener underflow	90.7%
% additional recovery of oil by No. 2 thickener	41.0 %
% indicated overall recovery by both thickeners from classifier overflow	95.8 %

4. — FLOW-SHEET OF A PROPOSED 500 TON/24 HOURS PILOT MILL

It is evident that commercial exploitation of bituminous sands is possible only on a large scale. Tonnages of not less than 10,000 per day have to be considered. However, the operation of the Mines Branch pilot plant cannot be used as a basis for the design of a 10,000 ton mill because it does not give sufficiently conclusive data on mill capacities for large-scale calculations.

It has been shown in the previous section, by using pebble charge as a basis for capacity computations, that a 9 ft. by 8 ft. low-discharge pebble-mill will have a capacity of 500 tons of bituminous sands per 24 hours. On the other hand, for commercial mills grinding quartz-gold or base metal ores, Lake Shore established the validity of Coghill's formula; namely, that the capacity and horsepower of a cylindrical mill vary as the diameter to 2.6 power. But whether Coghill's formula will apply when disintegrating the bituminous sands, which form such a sticky, viscous mass inside the mill, we had no means of determining. At the same time it was calculated, from the operating data furnished by the manufacturers who sell pebble-mills for grinding paints, that the capacities of mills used for that purpose vary as the diameter to 2.4 power. If this applies to bituminous sands then it can be calculated that a 7 ft. by 8 ft. low-discharge pebble mill has a capacity of 500 tons of bituminous sands per 24 hours.

Basing the horsepower requirements on pebble charge, at the rate of 1 h.p. per 300 lb. of pebbles for high-discharge mills and adding 30 per cent to this for low-discharge, the power required for a 9 ft. by 8 ft. mill will be 104 h.p. and that required for a 7 ft. by 8 ft. mill, 68 h.p. These figures check closely with that obtained from our mill runs in which a feed rate of 450 lb. of bituminous sands per hour was maintained.

From the foregoing it can be seen that before the design of a 10,000-ton unit is attempted, capacities and horsepowers of pebble-mills should be first established. This can be done by operating a 500 ton per 24 hours pilot unit, in which a 9 ft. by 8 ft. (or 8 ft. by 12 ft.) low-discharge pebble-mill should be used, though there is very little doubt in the writer's mind that a 7 ft. by 8 ft. mill, consuming only 68 h.p., will be quite adequate to treat 500 tons per 24 hours.

A 500-ton pilot unit will also supply the necessary information as to the sizes of classifiers and thickeners which will give the best oil extractions on a commercial scale.

A manufacturing company, basing their calculations on the results obtained by our pilot plant, specified the raking capacity of an 8 ft. *'F'* classifier as 20 tons of sands per foot of width in 24 hours, or a total of 160 tons per day, the rakes making 10 strokes per minute. This capacity might represent the economic efficient performance of the classifier and it might not. The operation of these classifiers in a 500-ton plant will give the required information.

Naturally, these classifiers should be equipped with variable speed drives so as to determine the best speeds for the maximum tonnages together with the most efficient extractions.

Basing thickener capacities for efficient handling of oil slurries from 500 tons of bituminous sands per 24 hours on the results obtained in the pilot plant, the dimensions of the primary thickener were calculated at 50 ft. by 8 ft. and of the secondary thickener at 40 ft. by 8 ft.

The above discussion outlines the essential points on which the design of the 500-ton flow-sheet shown in Figure 9 is based.

The cost of this equipment, together with motors, f.o.b. Waterways, Alberta, will be around $140,000. The total cost of the plant, erected on site, will be roughly $300.000. This will include the cost of the mill laboratory.

Personnel to operate this 500-ton plant on a 3-shift basis will be:

1 Mill superintendent
1 Assistant mill supt., to act also as mill statistician
1 Research engineer
3 Working shift bosses (optional)
3* Pebble-mill operators (who will also operate the agitator and classifiers)
3* Thickener operators
2 Repairmen on duty during day shift
1 Laboratory assistant

*These operators will also do all necessary cleaning in the mill.

Estimated costs for operating the plant are given in Table IX.

TABLE IX.— ESTIMATED MAXIMUM COSTS OF SUPPLIES, MAINTENANCE, AND REPAIRS PER TON OF BITUMINOUS SANDS, EXCLUSIVE OF SUPERVISION, OPERATING LABOUR, AND POWER

Belt feeder and screw conveyor	$0.004
Pebble-mill (pebbles, liners, lubrication, etc.)	0.073
Agitator	0.008
Classifiers	0.009
Thickeners	0.016
Pumps	0.020
Electrical equipment	0.020
Assaying	0.010
Miscellaneous	0.020
TOTAL for supplies, maintenance, repairs, etc.	$0.180

It must be noted here that the costs in this table are based on the assumption that the capacity of a 9 ft. by 8 ft. pebble-mill is 500 tons, as also is the combined capacity of three 8 ft. classifiers. Actually, if the capacities of these machines are higher, their operating costs will be correspondingly lower.

It is proposed to use Buhrstone (Belgian Silex) for lining the shell and feed end of the pebble-mill.

The materials used, and their cost, for a 9 ft. by 8 ft. mill are as follows:

	COST FOR 9′ BY 8′ MILL
Buhrstone	$1,028.50
Cement	127.50
Hardener	36.00
TOTAL COST	$1,192.00

The life of 2½ in. thick lining in an 8 ft. by 8 ft. mill, grinding sandstone, as given by the manufacturers, is from 14 to 16 months. Assuming that the life of this lining in the 9 ft. by 8 ft. mill will be 12 months, the cost per day will be $3.26, or $0.0065 per ton of bituminous sands. Since, for the discharge end of the mill, some alloy-steel grates must be used, which at the most will amount to another $.0065 per ton, the total cost of lining per ton will not exceed $0.013.

The cost of the pebble consumption per ton in the above estimate of costs was taken to be $0.0516. However, there is very little doubt that in actual operating practice this cost will be lower.

5. — EFFECT OF REDUCED WATER ADDITIONS ON BITUMEN RECOVERIES

To maintain the temperature of oil slurries at around 77°F., in the cold climate of northern Alberta, it might be necessary to heat the plant water to about 86°F. In large-scale operations this, naturally, means an added expense which might interfere seriously with the economics of the cold water separation of bitumen. Therefore, reduction in the amount of water used in the process would become a matter of primary importance.

Three mill runs were made under stabilized conditions, changing only the amounts of water added to the circuit.

Table X is an analysis of water additions for each individual run.

The results obtained with these several water additions, together with the results from the earlier series of four runs (see Table I), are shown in Table XI.

Studying Table XI we can see that:

(1) As the amounts of water added to the circuit were cut down, the recoveries of oil started falling off due to the combined effect of the increased oil contents of both classifier sands and No. 2 thickener underflow.

(2) For the highest recovery of oil, a water to feed ratio of 3:1 appears to be the optimum.

(3) In all the runs, a thickener overflow was produced which was a sufficiently high-grade crude oil to be fed directly into the refinery. However, though there were scarcely any variations in the oil contents, nevertheless, the amount of mineral matter in the thickener overflow, when the ratio of water to mill feed was as low as 1.8:1, was nearly doubled. This tended to indicate that with any more reductions in water additions, a point would be reached where a crude oil suitable

TABLE X.— ANALYSIS OF WATER ADDITIONS

	RUN No. 1	RUN No. 2	RUN No. 3
Bituminous sands treated lb.	1,140	1,140	1,140
Water content of classifier sands lb.	271	298	295
Water content of No. 1 thickener overflow lb.	107	95	91
Water content of No. 2 thickener underflow lb.	3,055	2,606	1,735
Total water in the circuit lb.	3,433	2,999	2,121
Less moisture content of bituminous sands lb.	8	11	13
Total water added to circuit lb.	3,425	2,988	2.108
Water added to tube-mill lb.	361	361	350
Water added to agitator lb.	1,485	1,389	1,063
Classifier sprays (by difference) lb.	1,579	1,238	695
Ratio: Tube-mill water: Feed	0.3:1	0.3:1	0.3:1
Agitator water: Feed	1.3:1	1.2:1	0.9:1
Classifier sprays: Feed	1.4:1	1.1:1	0.6:1
Total water: Feed	3.0:1	2.6:1	1.8:1

TABLE XI.— EFFECT OF DIFFERENT AMOUNTS OF WATER ADDITIONS ON RECOVERIES AND FINAL PRODUCT

	SERIES OF RUNS 4	RUN No. 1	RUN No. 1	RUN No. 1
Bituminous sands treated lb.	3,110	1,140	1,140	1,140
Feed rate per hour lb.	215.0	227.7	225.7	232.7
% bitumen in the sands %	15.9	15.5	14.4	15.4
Total bitumen in the sands lb.	494.5	176.7	164.2	175.6
Kerosene added lb.	482.0	181.8	181.8	176.4
Total oil lb.	976.5	358.5	346.0	352.0
Span 40 lb./ton	0.041	0.041	0.041	0.041
Soda ash lb./ton	1.51	2.0	2.0	2.0
Temperature of classifier overflow °F.	77.0	77.5	76.6	76.3
pH of classifier overflow	9.1	8.5	8.4	8.3
Ratio of total added water to feed	3,5:1	3:1	2,6:1	1.8:1
Weight of wet classifier sands lb.	2,811	1,124	1,112	1,119
% oil in classifier sands %	0.26	0.3	0.2	0.6
% classifier extraction %	99.3	99.1	99.3	98.1
Weight of No. 2 thickener underflow (tails) lb.	10,120	3,080	2,640	1,760
% oil in tails %	0.1	0.1	0.3	0.3
% recovery of oil by thickeners from classifier overflow %	99.0	99.1	97.7	98.5
% overall recovery of oil %	98.2	98.2	97.1	96.6
% actual recovery of bitumen %	96.4	96.4	94.2	93.2
No. 1 thickener overflow: % Oil %	75.6	75.2	76.6	75.7
% M.M. %	1.7	2.1	1.8	4.0
% H_2O %	22.7	22.7	21.6	20.3

TABLE XII.— EFFECT OF CLAY ON OIL SEPARATION AND RECOVERIES

SERIES OF RUNS	BITUMINOUS SANDS FEED, Lb.	CLAY IN FEED, lb.	CLAY IN FEED %	RATIO OF BITUMEN TO DILUENT IN MILL FEED	VISCOSITY OF OIL, Centipoises	WETTING AGENT USED	CLASSIFIER EXTRACTION %	LOSS OF OIL TAILS, lb.	CLAY IN TAILS, lb.	OIL LOSS PER LB. OF CLAY, lb.	OIL LOSS IN TAILS PER TON OF FEED, lb.	RECOVERY OF OIL BY THICKENER FROM C.O.F.	INDICATED RECOVERY %
MILL RUNS WITH KEROSENE													
1.....	6,550	269.3	4.1	51.2:48.8	103.7	Span 80	98.6	17.6	175.6	0.100	5.4	99.1	97.8
2.....	4,050	165.8	4.1	51.4:48.8	103.7	Rosin Amine D Acetate	98.6	18.5	112.3	0.165	9.1	98.5	97.1
3.....	6,600	281.2	4.3	48.3:51.7	103.7	Span 80	98.8	22.0	197.1	0.112	6.7	98.9	97.7
4.....	4,150	175.0	4.2	62.7:37.3	——	Span 80	98.7	19.8	120.9	0.164	9.1	98.2	96.9
5.....	3,110	149.4	4.8	50.8:49.2	103.7	Span 40	99.3	10.1	167.8	0.060	6.5	99.0	98.2
MILL RUNS WITH BITUMOUNT DILUENT													
6.....	4,450	195.7	4.4	50.4:49.6	336	Span 80	99.2	35.6	169.2	0.210	16.0	97.4	96.5
7.....	5,700	233.7	4.1	49.0:51.0	336	Span 80	99.0	27.3	106.3	0.257	9.6	98.6	97.5

for the refinery use could no longer be produced.

(4) Even with such a low ratio of water to feed as 1.8:1, the bitumen recovery obtained was 93.2 per cent. Until this is confirmed by the operation of a 500-ton plant, it might be said that the best economical balance would be obtained with water to feed ratio of somewhere between 2.5:1 and 2:1.

PART II. — ANALYSIS OF THE ROLE OF CLAY IN THE COLD WATER PROCESS

1. — *Effect of Clay on Oil Separation and Recoveries*

The bituminous sands treated in the pilot plant contained some clay and it was realized that practically all the clay in the feed would overflow the classifier and be pumped into the thickener.

After running the plant for four months it became apparent that:

(*a*) Silica sand is easily separated from oil and most of it can be rejected in the first stage of separation, namely, classification.

(b) Oil separates with difficulty from clay, and once the clay sinks to the bottom of a thickener or tank, it drags down the oil adhering to it.

These observations were similar to those made already by Clark and Pasternack in the hot-water separation process (4).

To make studies on the effect of clay it was decided that both the mill feed and classifier overflow should be analyzed for clay.

While the laboratory tests had already indicated that certain amounts of clay in the feed were necessary to effect better primary separations of oil, the causes of the effect which clay had on oil separations and recoveries were not understood. However, a suggestion was made which seemed to offer a reasonable explanation with respect to both. It has been thought that bitumen, being an asphaltic oil, has one, two, or more unsaturated hydrocarbon bonds which might have a greater tendency to be adsorbed on clay than saturated hydrocarbons. Thus, when the clay sinks to the bottom of the thickener, it drags down the oil adsorbed on it. By the same token, it can be expected that when there are certain amounts of clay in the feed, most of this clay, with oil adsorbed on it, overflows the classifier and leaves cleaner classifier sands behind.

Table XII is an analysis of the effects of clay on the separation and recoveries of oil.

From a study of Table XII, several conclusions can be drawn, as follows:

(1) As the clay content in the feed increased from 4.1 to 4.8 per cent, the classifier extractions, equivalent to oil separations, increased from 98.6 to 99.3 per cent. These extractions appear to be independent of oil viscosities and of the nature of wetting agents.

(2) Using the same wetting agent, with increase of clay in the thickener feed, the recoveries of oil in the thickeners were affected adversely.

(3) Using *Span* 80 with kerosene in series 1 and 3 and with Bitumount diluent in series 6 and 7, the amount of oil per lb. of clay in the thickener overflow, within the empirical limits of the tests, appears to be constant for each diluent. In other words, the loss of oil in the thickener per ton of feed is directly proportional to the amount of clay in the thickener underflow.

(4) The character of wetting agents appears to influence the oil loss per lb. of clay. The lowest loss was obtained with *Span* 40 and the highest with *Rosin Amine D Acetate*.

(5) With the increase in viscosity of oil, loss per lb. of clay appears also to increase. The increase in oil viscosity is due either to lesser amounts of diluent added or to the nature of the diluent itself.

2. — *A Tentative Study of Factors Causing Oil Losses*

Almost from the beginning of the investigation, it was evident that, by settling any kind of diluted crude oil, a product suitable for refinery treatment was obtained when most of the mineral matter and water were settled out. However, settling of thickener overflow containing some mineral matter, which was practically all clay, resulted in formation of sludge which settled at the bottom of storage tanks.

Tables XIII and XIV show the oil balances for 33 runs made with kerosene and 10 runs with Bitumount diluent, respectively, and illustrate the excessive losses due to sludge formation.

Since the recovery of oil locked in sludge turned out to be a very refractory problem, the formation of sludge became a nuisance and a source of excessive oil losses. Hence, all efforts were concentrated on obtaining a thickener overflow of sufficiently high grade to be fed directly into the refinery.

When this was finally achieved, settling was rendered unnecessary and formation of sludge was eliminated. Nevertheless, a study of sludge formation was made with a hope that it might lead to a better understanding of factors causing oil

losses in the cold water separation process.

It has been already suggested that the linking of oil with clay was due to adsorption. At the same time, if we consider the clay structure, it can be compared to a three-decker sandwich in which the bread is made of silica sheets (10) and interlayered in the upper and lower sheets by the non-exchangeable cations such as magnesium, aluminum, and iron. In between the upper and lower sandwiches is an area of loose bonding which, in effect, is the area of good cleavage of the clay minerals. Depending on the electronegativity of the silica sandwiches, this space is taken up by other cations, *e.g.*, H, Na, K, Ca, etc., and a certain portion of these are exchangeable, depending on their location within the layer (5, 9).

It has been found by experimental methods that with some clays, such as bentonites for instance, a certain percentage of these ions can be exchanged by other ions, depending on the valence and radius of the exchanging ion. Actually, large ions can be pushed into this space with a resulting expansion in one direction of the whole lattice. This has been found true for many organic ions as well (7).

If this were the only case, the problem, though still complex, would be comparatively simple. However, it has been found that neutral compounds (H_2O, etc.) can also be pushed into the structure, and it is believed that they settle into this area. What happens, in effect, is a polarization within the neutral compound, with the required charge moving to that portion of the compound where it is required for bonding. But is there a control over the amount of adsorption of neutral compounds, saturated hydrocarbons being one of them?

This adsorption phenomenon is evidently complicated by the external surface of the clay particle. It is quite obvious that the edges of the various layers of clay are ruptured, with open bonds existing as sites for ionic adsorption. Naturally, the smaller the clay particle, the larger the number of open bonds. Thus, the specific surface of clay particles becomes a very important factor in oil adsorption.

Evidently, different types of clays have different exchanging ions. However, for the purposes of our discussion we may assume, on the basis of the results of our mill runs, that Horse River bituminous sands contain the same type of clay with the same exchanging ions. We have

TABLE XIII.— OIL BALANCE WITH KEROSENE AS DILUENT
(35 Runs)

	LB.	%
Pure oil produced	11,211.0	87.23
Oil in samples (6 lb. per run, estimated)	198.0	1.54
Total oil in sludge	768.4	5.98
Oil in total tails	387.9	3.02
Assumed oil losses due to evaporation, spills, etc., plus discrepancies due to errors in sampling and assaying	286.8	2.23
Total = indicated total input of bitumen plus kerosene, basing this figure on assays of daily grab samples of bituminous sands and the measured amounts of kerosene added	12,852.1	100.00

TABLE XIV.— OIL BALANCE WITH BITUMOUNT DILUENT*
(10 Runs)

	LB.	%
Pure oil produced	2,800.1	85.51
Oil in samples (6 lb. per run, estimated)	60.0	1.83
Total oil in sludge	320.1	9.78
Oil in total tails	94.4	2.88
Total = indicated total input of bitumen plus diluent, basing this figure on assays of daily grab samples of bituminous sands and the measured amounts of diluent added	3,274.6	100.0

*There were no spills, and evaporation losses were assumed to be negligible because of the cold weather.

seen that, with increase in viscosity of the oil, the oil loss per lb. of clay also appears to increase. It is suggested here that the viscosity of oil influences the adsorption of hydrocarbon bonds. It might be added that the surface tension of the oil also influences the adsorption phenomenon. Thus, if we denote by Y the amount of oil in sludge per lb. of clay in sludge, we will have a system in which Y is dependent on:

μ, coefficient of viscosity
S, specific surface of clay particles
σ, surface tension of oil

which are all dimensional quantities.

But the relationship of Y to these dimensional quantities is complicated by the fact that Y also depends on the chemical constituents of bitumen, which are dimensionless quantities, and on the molecular weights and volumes of these constituents. Thus, it can be seen from the above discussion that the study of the factors causing oil losses in the cold water process poses a very complex research problem.

The complexity of this problem was emphasized and confirmed when a few mill runs were made on bituminous sands from the Bitumount property of the Oil Sands Project of the Alberta Government, on the northern extremity of the bituminous sands deposits. The Horse River sands came from the southern extremity.

The Bitumount sands contained more clay than the Horse River sands, so it was essential to ascertain the behaviour of increased amounts of clay in the cold water process. Since the amount of Bitumount sands received* was small, only a limited number of mill runs was possible and only one wetting agent could be tested.

The results obtained in 10 mill runs with Bitumount sands are summarized in the first column of Table XV. The second column in the same Table shows, for purposes of comparison, the results obtained with Horse River sands under similar conditions.

Studying Table XV we can see that, under similar conditions, the oil losses with Bitumount sands appear to be higher than those with Horse River sands.

From what was said before in this section, it can be stated that these higher oil losses with Bitumount sands can be ascribed to the changed natures of the clay and/or the bitumen. There is little doubt that this is so, for the following reasons:

*Received through the courtesy of Dr. Paul Gishler, National Research Council.

(1) The erratic and very often high *pH* values obtained during the mill runs with Bitumount sands not only indicated varying amounts of clay but also that these sands contained a clay or clays of different composition from that in Horse River sands.

(2) The lower viscosity and much higher fluidity of the approximately 50:50 mixture of Bitumount bitumen with kerosene definitely indicated that the chemical constituents of this bitumen were different from those of the bitumen from Horse River sands. Furthermore, the bitumen from the Bitumount sands appeared to be stickier and less easily wetted by soap. For instance, to remove the combined oil from one's hands two washings sufficed for Horse River oil, whereas not less than three vigorous scrubbings with soap were required for Bitumount oil.

(3) Consolidating all our observations, it might be said that the higher oil losses with Bitumount sands as compared with Horse River sands were principally due to the changed nature of the clay, which, in the former, had more exchangeable ions and possibly higher specific surface, and/or to the change in the nature of the chemical constituents of the bitumen itself.

(4) The thickener overflow with Bitumount sands was higher both in oil and clay than the thickener overflow with Horse River sands. This again would tend to indicate the different nature of the clay and/or the different nature of the bitumen in the Bitumount sands.

3. — *Comparison of the Oil Loss in Thickener Underflow with the Oil Loss in Sludge*

When sludge was dumped into barrels and stirred with an iron rod, some of the entrapped oil was liberated and was recovered. Eventually, a point was reached when no more oil could be recovered by further stirrings, and the sludge appeared to be a stable emulsion. In experimenting with one particular sludge which resulted from settling when *Span* 80, for the most part, was used as the wetting agent, it was found that the amount of oil to 1 lb. of clay was 0.82 lb. On the other hand, as shown by Table XII. the loss of oil per lb. of clay in the thickener underflow when *Span* 80 was used varied from 0.100 to 0.112 lb. In both cases, kerosene was used as a diluent.

From observations made on the small portion of clay from the thickener feed which remained in the thickener overflow, it was obvious that it caused sludge formation when the overflow was settled and that it was much more finely subdivided than the clay that settled into the thickener underflow.

When the pilot plant runs were being completed, no direct tests were made to size the clay in the thickener underflow and the sludge. However, a simple reasoning, based on observations on times of settlement and occasional screen analyses of mineral matter in the thickener feed, indicated that the average size of clay particles settling into the thickener underflow was roughly between 70 and 80 microns, and that of the sludge was probably somewhere between 14 and 8 microns. In other words, the ratio of the average particle size in the thickener underflow to the average particle size in the sludge varied somewhere between 5 to 1 and 10 to 1. From this it could be calculated that the specific surface of the clay particles in the sludge would be 5 to 10 times greater than that of the clay particles in the thickener underflow, or on an average 8 times greater. Since the respective oil losses per 1 lb. of clay in the sludge and thickener underflow were 0.82 lb. and 0.10 (or 0.112) lb., it appears that the amount of oil adsorbed on the clay was, within the limits of correctness of our deductions regarding particle sizes, directly proportional to the specific surface.

TABLE XV.— RESULTS OBTAINED WITH BITUMOUNT BITUMINOUS SANDS

	BITUMOUNT SANDS	HORSE RIVER SANDS
Bituminous sands treated ..... lb.	11,070	3,110
Rate of feed ..... lb./hr.	231.3	215.0
% Bitumen in the sands ..... %	15.1	15.9
% Clay in the sands ..... %	6.6	4.8
Soda ash ..... lb./ton	2.50	1.51
Span 40 ..... lb./ton	0.047	0.041
Average temperature ..... °F.	77.2	77.0
Average *pH* of classifier overflow ..... *pH*	9.5	9.1
Variations of *pH*	8.7 to 11.7	9.05 to 9.2
Ratio of bitumen to kerosene in feed	49.3:50.7	50.8:49.2
Viscosity of oil, centipoises at 70°F	31.4	103.7
Specific gravity of oil at 70°F	0.895	0.915
% M.M. eliminated by classifier ..... %	85.5	82.0
% oil in classifier wet sands ..... %	0.45	0.26
Weight of classifier wet sands ..... lb.	10,450	2,811
% extraction of oil by classifier ..... %	98.6	99.3
Oil in classifier overflow ..... lb.	3,341.1	974.8
Clay in classifier overflow ..... lb.	295.5	189.5
% oil in No. 2 thickener underflow (tails) ..... %	0.32	0.10
Tails to waste ..... lb.	32,280	10,120
Loss of oil in No. 2 thickener underflow ..... lb.	120.6	10.1
Clay in No. 2 thickener underflow ..... lb.	190.7	167.8
Oil loss per lb. of clay ..... lb.	0.63	0.06
Oil loss per ton of feed ..... lb.	21.8	6.5
% recovery of oil by thickeners from C.O.F.	96.4	99.0
% indicated overall recovery ..... %	95.0	98.2
Thickener overflow: % Oil ..... %	80.4	75.6
% M.M. ..... %	2.6	1.7
% H_2O ..... %	17.0	22.7

This is a confirmation of the fact that the adsorption of oil on clay is a function of the specific surface of the clay particles.

It might be added that the formation of a stable emulsion also appears to confirm the effect of the surface area of clay particles. There is a well-founded theory (8) which explains how fine insoluble particles can stabilize emulsions of water in oil. This is a surface phenomenon, and depends on the tendency of oils to wet a surface. The clay particles, having adsorbed the oil, are wetted by it and so cause the emulsion of water in oil to become stable.

This might indicate that, if specific wetting agents can be found which would cause the clay particles to be wetted mostly by the water, then the oil losses in the cold water separation process, if not entirely eliminated, might be appreciably reduced.

SUMMARY AND CONCLUSIONS

(1) A flow-sheet with standard ore dressing machinery was developed which, using Bitumount diluent and small amounts of soda ash and wetting agent, obtained an actual recovery of 95 per cent of bitumen from Horse River sands.

(2) The use of soda ash is imperative. When its use was discontinued, the result was always low recoveries of oil, even if the *pH* was over 9.

(3) The optimum range of temperature was found to be between 73° and 81°F., though good recoveries were obtained at as low a temperature as 70°F. and as high as 83°F.

(4) The optimum *pH* required appeared to be just over 9.

(5) To obtain the best recoveries, the mineral matter in the sands had to be reduced in the pebble-mill from 5 to 7 per cent minus 200 mesh to 11 to 15 per cent minus 200 mesh. This grind was the measure of efficient loosening and scouring of the bitumen off the sands.

(6) The increase in clay content affected thickener recoveries adversely. However, since the best separation of oil in the classifier took place with a certain content of clay in the mill feed, there appears to be an optimum content which will assist in effecting the highest overall recoveries.

(7) Crude oil of a grade suitable for refinery treatment was produced consistently in a continuous process.

(8) Pilot plant runs on cold water separation of bitumen were carried out as pure ore dressing investigations. However, on the completion of these investigations, a research problem was posed, which, it is hoped, will be taken up by future investigators.

(9) From the results obtained, a flow-sheet for a 500-ton per 24 hours mill has been worked out. It is suggested that this be considered as a 500-ton pilot mill on which the design of a 10,000-ton unit can be based.

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Metallurgical Practice at Golden Manitou

By D. A. LIVINGSTONE *

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ABSTRACT

Golden Manitou Mines, Limited, nine miles east of the town of Val d'Or in northwestern Quebec, are presently milling, at the rate of 1,000 tons daily, a relatively complex sulphide ore and producing, in decreasing order of value, zinc, silver, gold, lead, and copper. The products are a zinc concentrate, a silver-lead-copper concentrate, and a silver-gold precipitate from cyanidation.

Although several independent pre-production metallurgical investigations into the treatment of the ore were carried out, current practice is different from any of the recommended methods and, while still leaving room for improvement, is considerably more effective than earlier performance.

Flow-sheet developments from the commencement of operations in 1942 are dealt with briefly, showing the reasons for making numerous changes, the shortcomings of the various schemes tried, and, finally, covering data pertinent to present-day practice. From the recovery standpoint, silver has presented the greatest difficulty. The writer's association with the Company coincided with the start of production.

Characteristics of the Ore

THE PREDOMINANT metallic minerals present in the ore, in decreasing order of their abundance, are: pyrite, sphalerite, galena and chalcopyrite, with lesser amounts of argentiferous tennantite-tetrahedrite, arsenopyrite, covellite, bornite, and chalcocite. The sphalerite is mostly light in colour, sometimes colourless, and practically free from combined iron. Some of the sphalerite is very closely associated with minor amounts of pyrite, chalcopyrite, and galena. Pyrite is abundantly disseminated as medium to fine grains; more rarely it is massive and contains occasional grains of sphalerite, galena, and chalcopyrite. Chalcopyrite and galena occur as medium to fine irregular grains associated with the other sulphides.

Concerning the precious metals, visible gold is extremely rare. However, assays indicate that the gold is associated with chalcopyrite, pyrite, and sphalerite — predominating in the copper and iron sulphides. In addition to argentiferous tennantite-tetrahedrite, a portion of the silver occurs in the native state as particles partially coated with a soft grey material which has not been identified. On rare occasions, wire silver has been found in the underground workings, and some electrum has been noted. Frequent microscopic examinations, both of the ore and of mill products, have failed to reveal the complete picture of the association of the silver.

The gangue consists of fine textured milky quartz, sericitic and chloritic material, and carbonates.

The metal content of the ore has varied over a wide range but the mill heads for the past eighteen months have averaged as follows:

Gold	0.037 oz./ton
Silver	4.28 "
Zinc	3.82 per cent
Lead	0.50 "
Iron	7.00 "
Copper	0.15 "
Arsenic	0.03 "

Pre-Production Metallurgical Test Work

Samples of ore were submitted for test work to several qualified metallurgical laboratories, with a view to the selection of a suitable mill flow-sheet. Roughly speaking, three alternative flow-sheets were suggested, as follows:

A.—A complete sulphide float (bulk float), for cyanidation with a zinc float on the cyanide residue; bulk float tailing and zinc float tailing to waste.

B.—A differential lead-zinc-iron float followed by cyanidation of the combined lead and iron concentrates.

C.—Cyanidation of total mill feed followed by a zinc float on the cyanide residue (Figure 1).

Flow-sheet *B* appeared to be the least desirable owing to indicated high losses of zinc in the lead and iron floats and to precious metal losses in the zinc concentrate.

Of the remaining choices, flow-sheet *C* (Figure 1) appeared, on the basis of laboratory work, to be preferable in that it indicated better precious metal recoveries and less flotation equipment. Accordingly, this flow-sheet was adopted for a start but, in consideration of the optional methods of treatment, provision for reasonable flexibility of the flow-sheet was incorporated in the purchase and installation of equipment.

Flow-Sheet Development

From the outset, the mill operation was beset by numerous difficulties, both mechanical and metallurgical. It quickly became apparent that flow-sheet changes would be both necessary and desirable. A discussion of current metallurgical practice will be more significant if the flow-sheet changes leading up to it are dealt with chronologically.

Flow-Sheet No. 1

Essentially, flow-sheet No. 1 (Figure 1) consisted of cyanidation of the total mill feed, followed by flotation of the zinc.

At an optimum solution strength equivalent to 3.0 to 3.5 lb. KCN per ton of solution, recovery figures for gold and silver were respectively 75 per cent and 55 per cent, which were a little lower than predicted. Zinc concentrate grade was quite good, namely 57 per cent, but recovery was disappointing, being of the order of 75 per cent. The low recovery was believed to be due to the relatively long

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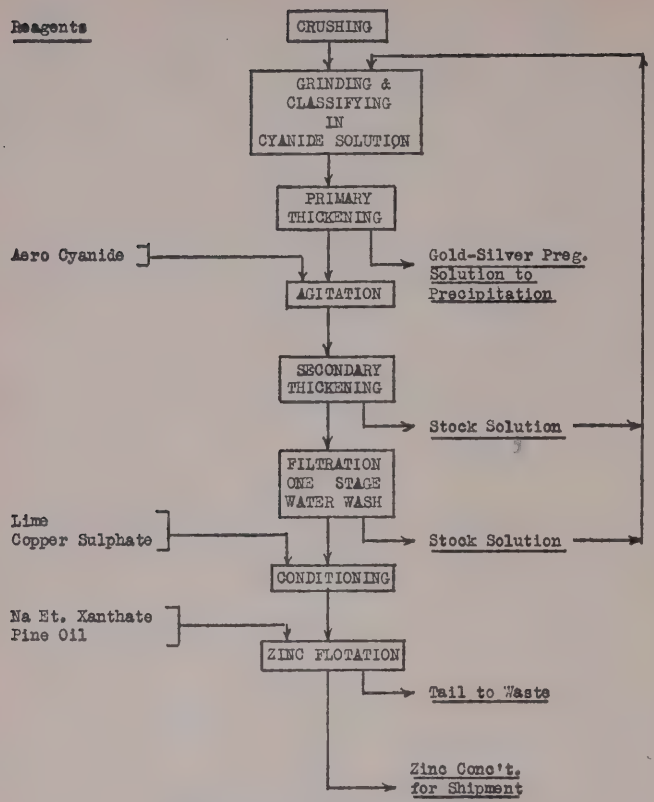

Figure 1.—Flow-sheets C and 1.

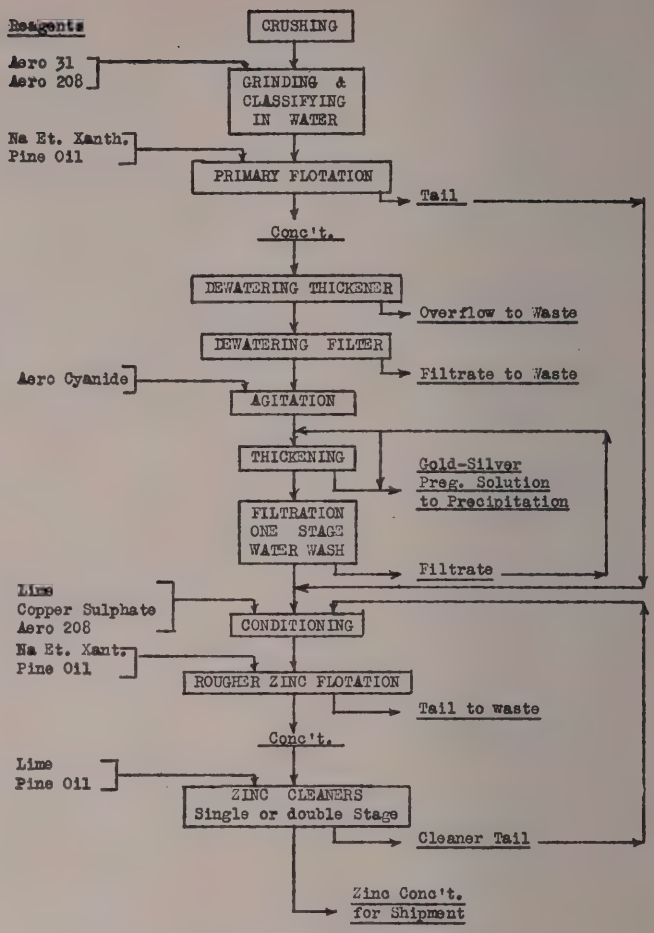

Figure 2.—Flow-sheet 2.

exposure of the sphalerite to the depressing action of cyanide. Moreover, this situation was aggravated by the inefficient washing of the cyanide residue on disc-type filters, resulting in abnormally high copper sulphate consumption to activate the sphalerite. It was further noted that, as the reducing power of the solutions in the cyanide circuit increased, there followed a decrease in zinc flotation recovery. This behaviour was not satisfactorily explained.

After operating this flow-sheet for approximately two months at a reduced rate of 300 tons daily, it became apparent that the thickening capacity provided was inadequate for higher tonnage. Various settling agents were tried, to no avail. Due to commitments for zinc concentrate output, it was imperative to step up to the originally planned milling rate of 600 tons daily. To do this it became necessary either to double the thickening capacity in the cyanide circuit or to alter the flow-sheet to allow for the treatment by cyanidation of only a portion of the tonnage milled. The first alternative was ruled out because of the delay and expense involved in new equipment and a building extension.

Flow-Sheet No. 2

Flow-sheet *A*, suggested by the preliminary test work, offered a method of reducing the tonnage to be cyanided but, on the other hand, would not eliminate the undesirable feature of flow-sheet No. 1, namely, the necessity of floating zinc from cyanide residue alone. For this reason, a modification of flow-sheet *A* was adopted at this point (see Figure 2). Instead of a complete sulphide float, test work indicated that, by making a partial sulphide float with no attempt to activate zinc, reasonably good precious metal recoveries could be obtained in this type of primary concentrate, which consisted chiefly of pyrite. It was estimated that approximately 70 per cent of the zinc would remain in the primary tailing. This scheme necessitated both the primary tailing and the cyanide residue being floated for zinc. While this required greater cell capacity for zinc flotation, it offered improved efficiency in this circuit due to the fact that only a portion of the zinc heads consisted of cyanide residue.

With this flow-sheet (No. 2), the ratio of concentration in the primary flotation circuit was of the order of three to one. This meant that the cyanide circuit handled only 200 tons per day in place of the full tonnage as in flow-sheet No. 1, thus overcoming the thickening bottle-neck. Soon after starting this flow-sheet, however, an unforeseen difficulty was encountered; the arsenic content of the zinc concentrate increased from roughly 0.04 per cent to 0.09 per cent. It was urgently requested by the zinc smelter that the arsenic be lowered to 0.05 per cent in order to ensure a continued outlet for our zinc product. Furthermore, the primary concentrate, with a specific gravity of 4.2, led to mechanical difficulties in the cyanide circuit, where no provision had been made for the handling of this relatively coarse, heavy material. Due to the extremely rapid settling rate, difficulty was experienced in keeping the solids in suspension in the agitators, resulting in frequent disruption of operations. The increased tonnage rate was attained, and zinc recovery improved, but overall precious metal recoveries slipped somewhat.

This flow-sheet was operated for only four weeks. Because of the metallurgical and mechanical problems mentioned, a further change in flow-sheet was necessary.

Flow-Sheet No. 3

As expected, most of the arsenic floated with the pyrite in the primary flotation circuit and entered the zinc circuit with the cyanide residue, and

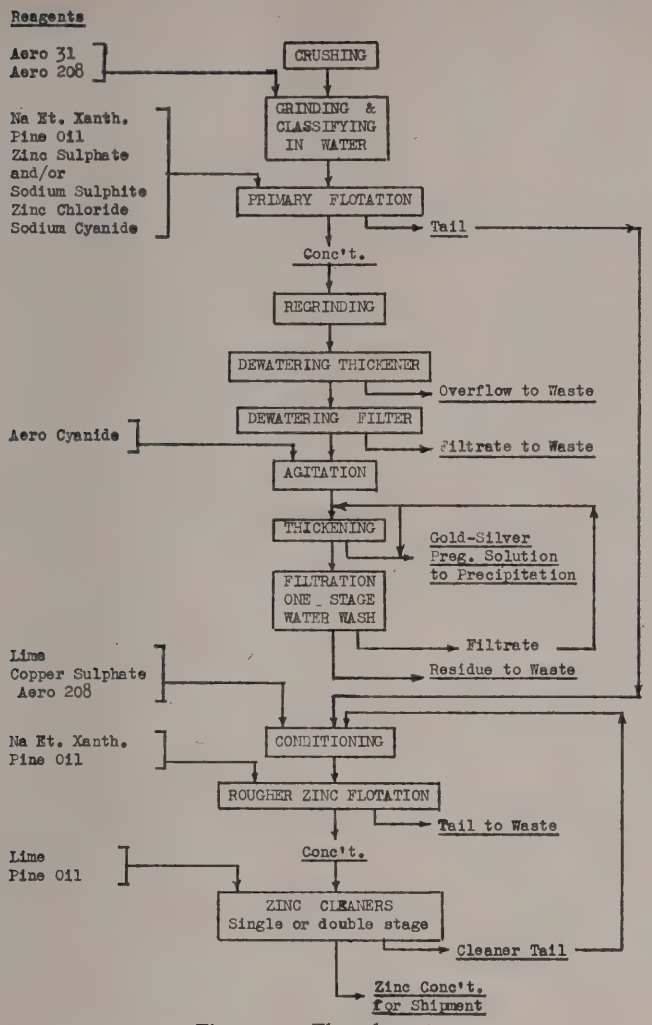

Figure 3.—Flow-sheet 3.

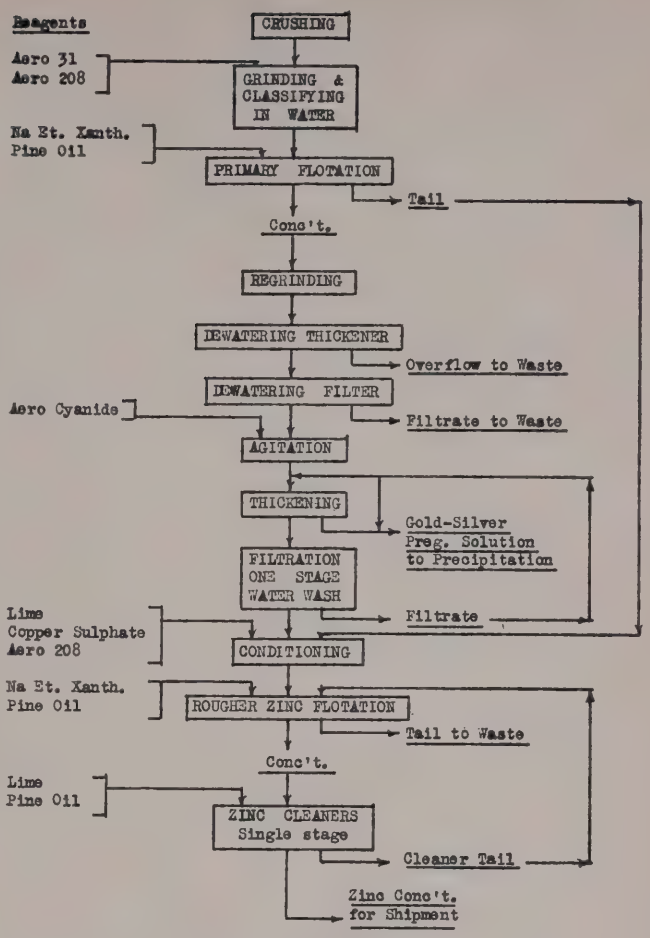

Figure 4.—Flow-sheet 4.

therefore the most immediate method of eliminating the arsenic from the zinc concentrate was to discard the cyanide residue instead of returning it to zinc flotation. In order to do this it became necessary to depress as much of the zinc as possible in the primary flotation circuit, to minimize the zinc lost in the cyanide residue. At the same time, in order to overcome the difficulty of handling a coarse, heavy concentrate in the cyanide circuit, re-grinding of this material was adopted. As a result, the fineness of the feed to the cyanide circuit increased from 60 per cent to 90 per cent minus 200 mesh. This, together with the installation of three auxiliary outside airlifts to each agitator, served to overcome the problem of handling this material in the thickeners and agitators.

The switch to this flow-sheet (Figure 3) reduced the arsenic in the zinc concentrate to a satisfactory figure, but the problem of depressing sufficient zinc in the primary flotation circuit proved to be a stumbling block. Numerous zinc depressants were tried. These included zinc sulphate, sodium sulphite, zinc chloride, and sodium cyanide, which were used in varying amounts and combinations. About this time microscopic examination established that the primary concentrate contained considerable pale yellow and pure white sphalerite which it was almost impossible to depress in the primary float. In actual practice, this loss amounted to the high figure of 17 per cent of the total zinc in the mill feed. The metal recoveries for the six-week period of operation were as follows:

Gold recovery by primary flot'n...	90%
" " by cyanid'n of primary conc.	86%
" " overall.....	77%
Silver recovery by primary flot'n..	85%
" " by cyanid'n of primary conc.	40%
" " overall.....	34%
Zinc recovery.....	80%
Zinc conc. grade (containing <0.03% As).....	55.7%

Failure to depress sufficient zinc, therefore, was the greatest objection to this flow-sheet. This in itself was the reason for reverting to a flow-sheet similar to No. 2 and seeking other methods to overcome the problem of excess arsenic in the zinc concentrate.

Flow-Sheet No. 4

Flow-sheet No. 4 (Figure 4), which went into operation about the end of the year 1942, was essentially the same as No. 2, except that the primary concentrate was now reground before going to the thickener. Moreover, the second agitator was put into operation in the cyanide circuit, after being out of service for some time due to an inadequate air supply.

It was found that the arsenic content of the zinc concentrate remained below the desired limit of 0.05 per cent. This was an interesting development, as it was not possible to attain this goal when operating previously under the almost identical flow-sheet No. 2. While it must be acknowledged that no positive explanation can be set down at this time to account for the solution of the arsenic problem, the following factors were thought to have a bearing on the matter:

(*a*) Re-grinding of the primary concentrate, thus liberating any attached arsenic minerals.

(*b*) Additional agitation time in the cyanide circuit. This may have conditioned some of the arsenic minerals, rendering them subsequently less floatable in the zinc circuit.

(*c*) Improved alkalinity control in zinc flotation resulting in better elimination of extraneous sulphides, including some of the arsenic minerals.

About this time, also, the zinc flotation circuit was altered to give more conditioning time by returning the zinc cleaner tailing to the head of the roughers instead of to the conditioner. This resulted in an improvement in zinc circuit performance and, combined with close alkalinity control, led, over a period of roughly two months, to an increase in zinc recovery from 80 per cent to 90 per cent. Precious metal recoveries for this period were of the order of 78 per cent for gold and 41 per cent for silver, when treating slightly more than 600 tons daily.

Early in 1943 it was decided to increase the milling rate to 1,000 tons per day. It was therefore necessary to select from the various flow-sheets, either proposed or tried, one which would yield the most satisfactory metallurgical results commensurate with a reasonable capital expenditure.

Flow-sheet No. 1, in spite of the higher silver recovery, was rejected as a possible choice because the thickening, agitating, and filtering capacities in the cyanide circuit would have to be increased by at least 300 per cent, involving a costly building addition.

Flow-sheet No. 3 was also rejected, due to excessive loss of zinc in the cyanide residue. It was therefore decided to continue with flow-sheet No. 4, in spite of low silver recovery, especially since it would require a minimum of new equipment and no addition to the building. Typical performance data for this flow-sheet, covering the ensuing five-month period, were as follows:

Gold recovery by primary flot'n	92%
" " by cyanid'n of primary conc.	83%
" " overall	76%
Silver recovery by primary flot'n	88%
" " by cyanid'n of primary conc.	47%
" " overall	41%
Zinc recovery	92%
Zinc. conc. grade	55.8%
Mill heads: gold, 0.053 oz./t.; silver, 3.27 oz./t.; zinc, 7.1%	

As a matter of interest it might be stated that the primary flotation cells were supercharged with air at one pound pressure during this period and by so doing there was effected a saving in power and frothing reagent of approximately 15 per cent. No appreciable changes in recoveries were noticed.

In an attempt to improve the silver recovery, test work in the mill laboratory was pursued along the following lines:

(1) To investigate all possible angles of the cyanidation treatment with a view to improving silver extraction. Broadly speaking, this work was intended to reveal the effects on silver extraction of the following factors and reagents: cyanide solution strength, type of cyanide, alkalinity, fouling of solutions, lead salts, mercury salts, sodium chloride, calcium hypochlorite, additional aeration and preaeration — with and without lime — agitation time, fineness of grind, roasting ahead of cyanidation, washing of the concentrate with caustic soda, sodium sulphide, ferric sulphate-sodium chloride mixture, and sulphuric acid to break down the silver minerals. It is not considered to be within the scope of this paper to discuss the results of this rather extensive laboratory investigation. However, from this work it was concluded that, beyond a slight increase in extraction from additional agitation time, for which no space was available, no economical means for improving the silver recovery by cyanidation of the ore then being treated had been discovered. This being the case, attention was directed toward a new field of endeavour.

(2) To investigate alternative flotation schemes, either to replace or supplement the cyanidation circuit. Two methods seemed worthy of study:

(*a*) A zinc float only, with depression of the iron, followed by re-grinding and cyanidation of the zinc cleaner concentrate. Using this method, the cyanide residue would be the final zinc concentrate. This suggested one advantageous feature in that the tonnage of feed to cyanidation would be reduced to one-quarter the amount then being cyanided, thereby increasing agitation time and improving filter efficiency, etc. Unfortunately, tests showed that only 50 per cent of the precious metals floated with the zinc. The remainder, most of which was associated with the pyrite, was lost in the zinc tailing, thereby lowering instead of raising precious metal recoveries.

(*b*) A lead float to remove, in a marketable lead concentrate, that portion of the silver which is not amenable to cyanidation. Owing to the low lead content of the ore — approximately three-quarters of one per cent — its value from the standpoint of lead alone was not great and hence more pressing metallurgical problems had not permitted thoroughly investigating its recovery.

It was established that, of the total lead and silver in the mill feed, approximately 60 per cent of the lead and 30 per cent of the silver floated with the zinc concentrate. Since no payment was received for lead and silver, their presence in this concentrate not only represented metal losses but also detracted from the grade of zinc concentrate. Thus any scheme to remove the lead offered three advantages: the recovery of silver would be materially improved, a lead return would be realized, and the grade and therefore the value of the zinc concentrate would be enhanced.

Study of this problem suggested that the lead and silver minerals might be removed at any one of four points in the mill circuit:

(1) From the new feed, which would entail lead flotation on the entire mill tonnage, followed by the reactivation and flotation of pyrite for cyanidation. This was discarded due to the great amount of additional space and equipment required.

(2) From the filtered cyanide residue, preceding zinc flotation. This would necessitate the treatment of 300 to 400 tons daily, from which the silver amenable to cyanidation had been removed. However, laboratory flotation work on cyanide residue from the mill circuit failed to produce acceptable results.

(3) From the zinc concentrate. In this case, a relatively small quantity, not exceeding 100 tons daily, would have to be handled. Laboratory tests showed that satisfactory recoveries of lead and silver could be expected by de-leading the zinc concentrate but, at the time, it was thought that a thickening and filtering stage, requiring a sizeable building addition, would be needed to effect suitable zinc depression.

(4) From the primary concentrate, preceding cyanidation. This method appeared, at the time, to possess the following advantages:

(*a*) Chalcopyrite and other copper minerals behaving as strong cyanicides would be eliminated from the cyanide circuit.

(*b*) Lead would be eliminated from the circulating load in the zinc cleaner circuit.

(*c*) A minimum of additional equipment and mill alterations would be involved.

Laboratory tests indicated that lead flotation at this point could be

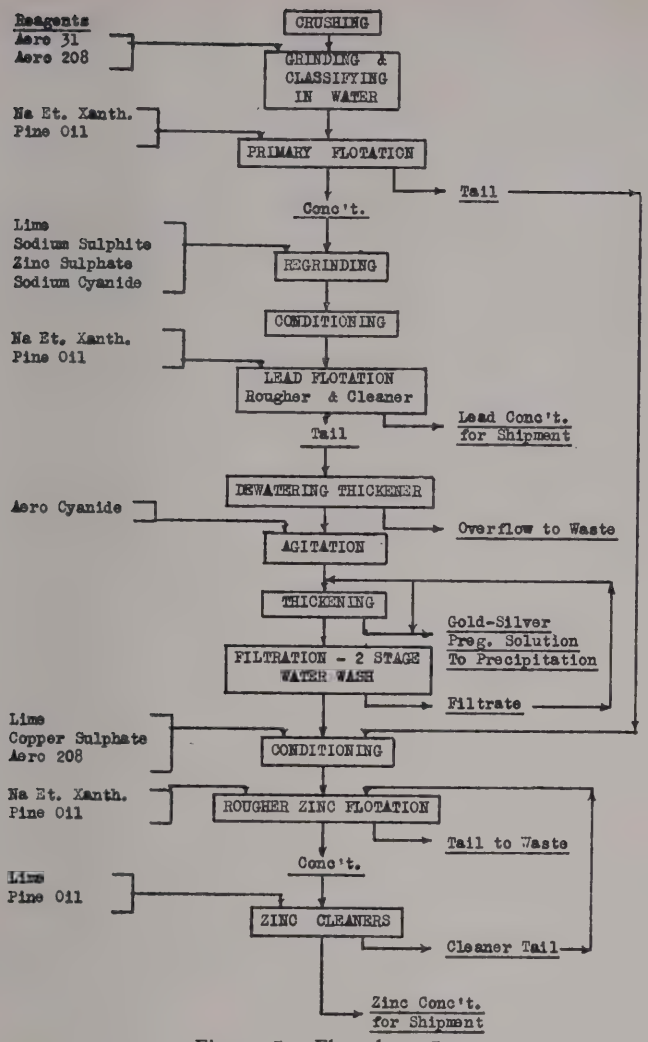

Figure 5.—Flow-sheet 5.

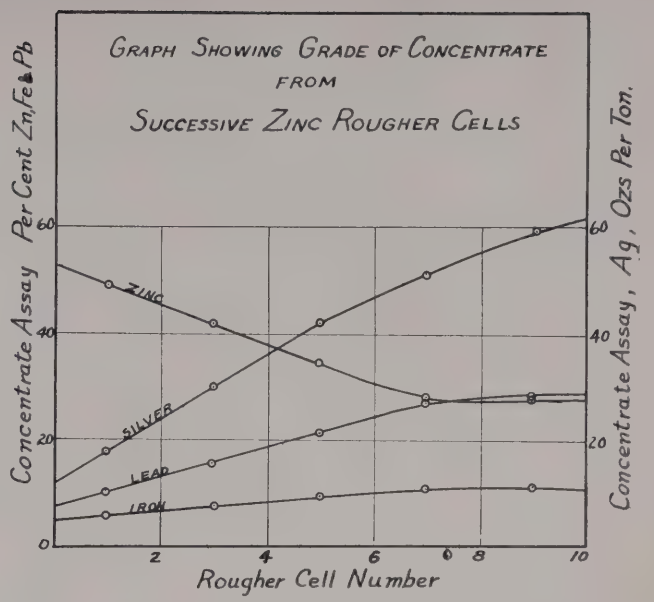

Figure 6.—Showing the relatively slow floatability of silver and lead compared with zinc, in the zinc roughers.

carried out by conditioning for 45 minutes, using lime, sodium sulphite, and sodium cyanide as iron and zinc depressants, followed by flotation using starvation amounts of xanthate. It was estimated that precious metal recoveries would then amount to 50 per cent of the total contents of the mill feed. This, of course, would be augmented by additional extraction in subsequent cyanidation, to boost the overall silver recovery from 41 per cent — then being obtained in the plant — to 66 per cent. Thus flow-sheet No. 5 (Figure 5) came into use.

Flow-Sheet No. 5

At the start, operating conditions in the plant were made to conform as closely as possible with those indicated by laboratory work. Results, however, failed to duplicate laboratory figures. With the exception of zinc rejection in the lead circuit, which was acceptable, and the reduction in cyanide consumption, which measured up to expectation in the cyanide plant, all other features of the undertaking were disappointing. Recovery of silver in the lead concentrate was down 15 per cent, and the ratio of concentration was too low, due to an excessive amount of iron (20 per cent) in the lead concentrate. This latter fault was especially undesirable because of the high freight charges to the lead smelter in Montana. Moreover, from the viewpoint of the operator, the control of the circuit was most difficult and results from hour to hour were very erratic. Despite the shortcomings of the flow-sheet there was, nevertheless, a profit from its operation. This profit was, however, somewhat less than half of that anticipated.

The reason why plant results for the de-leading of primary concentrate did not measure up to the laboratory tests could not be fully explained. The flow-sheet remained in operation for better than six months, during which time a great deal of experimental work was done. Conditioning time, densities, reagent rates and their points of addition, various reagent combinations, and the effect of temperature, were investigated. Single and double cleaning of the lead concentrates were also tried. From this work the following conclusions were drawn:

(*a*) That the pyrite, having been floated in the preceding circuit, required such high lime alkalinity for depression that the flotation of the lead and silver minerals was partially inhibited.

(*b*) That soda ash was not a suitable substitute for lime as an iron depressant on this ore.

(*c*) That sodium sulphite and zinc sulphate at 0.75 lb. of each per ton, and sodium cyanide at 0.05 to 0.10 lb. per ton, gave satisfactory zinc rejection.

(*d*) That the addition of depressing reagents to the re-grind ball-mill gave results superior to those obtained when adding them to the conditioner.

(*e*) That sodium cyanide at the rate of one pound per ton gave the best zinc depression but caused dissolution of about 20 per cent of the gold in conditioning.

Early in 1944, attention was diverted from the primary concentrate de-leading circuit by a problem of more immediate concern. At this time, a complaint was received from the zinc smelter regarding an excessive amount of insoluble matter (silica and alumina) in the zinc concentrate. After experimenting briefly, without success, using gangue depressants such as sodium silicate, caustic-starch, Cyanamid Reagent 610, and Aerosol in the zinc circuit, it was concluded that the zinc cleaner capacity was inadequate. For this reason, lead flotation was discontinued, in order to release the flotation cells for additional zinc cleaners. This move therefore meant a return to flow-sheet No.

4 (Figure 4), except that double filtration of cyanide residue had been adopted, at the expense of dropping a filtering stage ahead of cyanidation. The reduction of the amount of insoluble matter in the zinc concentrate from 6.5 per cent, being obtained at the time, to 3.0 per cent and less was readily effected and this improvement was attributed to the following corrective measures:

(*a*) Tripling the zinc cleaner cell capacity, thereby effecting an increase from single-stage to three stages of cleaning.

(*b*) Raising the *pH* in the zinc cleaners from 11.6 to 12.1, and that of the zinc flotation heads from 9.5 to 10.0.

(*c*) Reducing the copper sulphate feed rate to a minimum based on the zinc content of the heads as indicated by control assays.

(*d*) Closer control of the circulating load in the cleaners, guided by the assays and densities in the circuit.

(*e*) Doubling the conditioning capacity to make a more uniform mixture of primary tailing and cyanide tailing for feed to the zinc circuit. This was deemed advisable due to interruptions in the flow of cyanide tailing occasioned by changing of filter sectors, etc.

The De-Leading of Zinc Concentrate

What was gained in zinc concentrate grade by eliminating a portion of the insoluble matter was unfortunately offset by an increase in the lead content of the zinc concentrate, due to the discontinuation of the lead flotation circuit some months earlier. In addition, the silver recovery was once more down to the disappointing figure given for flow-sheet No. 4, namely 41 per cent, and on occasions it dropped as low as 35 per cent. Obviously, the solution for the above two problems, or at least a step in that direction, lay in some method of lead removal. As all available mill equipment was now being used, this meant that more equipment, and possibly a building extension, would be needed for any further additions to the flow-sheet.

At this time, further consideration was given to the previous laboratory work referred to earlier in this paper, to de-lead the zinc concentrate. In the light of the operating experience gained from de-leading the primary concentrate in flow-sheet No. 5, the de-leading of the zinc concentrate now appeared to have the following new advantages not previously fully appreciated:

(*a*) Since iron, as pyrite, amounted to only 2 per cent in the zinc concentrate, compared with 34 per cent in the primary concentrate, it would not be necessary to eliminate it in de-leading the zinc concentrate. This obviated the necessity of using lime as an iron depressant as required in the previous de-leading circuit (Figure 5), and consequently aided the flotation of the lead and copper-silver minerals.

(*b*) Since the tonnage of zinc concentrate to be de-leaded would be only about one-quarter of the primary concentrate de-leaded in flow-sheet No. 5, the conditioning and flotation cell requirements, and probably the reagents, would be greatly reduced.

Earlier laboratory work along these lines was confirmed and considerably improved by extensive tests in an improvised pilot plant. Based on this work, the de-leading of zinc concentrate was selected as the most feasible scheme for increasing the overall silver recovery, and for raising the grade of the zinc concentrate by eliminating the lead. The necessary equipment was then crowded into the existing building, in the interests of time and economy, and indicated advantages of a thickening and filtering stage ahead of the de-leading circuit were foregone owing to space limitations. The test work, installation, and evaluation of this de-leading circuit have been covered in an earlier paper (1) and there fore are not repeated here.

From the outset, the metallurgical performance of this addition to the mill flow-sheet was very satisfactory. Briefly, the chief benefits were: an increase in zinc concentrate grade from 57 per cent to 60 per cent; an improvement in the overall recovery of silver from 41 per cent to 61 per cent; a lead recovery amounting to approximately 48 per cent of that in the original mill feed. This method of treatment resulted in a loss of about 2 per cent of the zinc in the lead concentrate, but the increased revenue from the higher grade of zinc concentrate more than compensated for this loss.

(1) Livingstone, D. A., *The De-Leading of Zinc Concentrate at Golden Manitou Mines;* C.I.M. Bulletin, June, 1946, pp. 362-366.

Scavenger Flotation

With the incorporation of the de-leading addition in September, 1945, the operation of the mill, both mechanically and metallurgically, became, for the first time, sufficiently stabilized to permit proper metallurgical control and the accumulation of reliable data. Recoveries at a milling rate of 1,000 tons daily under these conditions are summarized in Table I.

Though markedly improved results were thus obtained, losses of silver, zinc, and lead in the zinc flotation tailing were quite appreciable, even though the theoretical contact time in the zinc rougher flotation section was calculated to be 16 minutes, or more than three times the requirement indicated by laboratory tests. It seems that this apparent discrepancy between indicated and actual capacity requirements has been the experience of numerous other zinc flotation operators in the industry.

Table I.— Recovery Data

	Gold	Silver	Zinc	Lead
Head assays	0.035 oz./ton	3.61 oz./ton	4.6%	0.7%
Recovery by cyanidation	74.7%	40.9%	—	—
Recovery by lead flotation	1.4%	20.1%	—	47.8%
Recovery overall	76.1%	61.0%	85.6%	47.8%

It had been known for some time that the zinc cleaner circuit rejects returning to the roughers were high in lead and silver values, thus adding to the burden on this circuit. Contributing to this condition was the depressing action of the high-lime alkalinity (*pH* 12) required to maintain good grade. The relatively slow floatability in the zinc roughers of the lead and silver compared with zinc is illustrated by the curves in Figure 6, derived from the sampling of individual cells. These emphasize the appreciable metal values, particularly silver, being recovered from the last cell, and strongly indicated the advantages that should accrue from longer flotation time.

Actually, this thought was not by any means a new one at this stage but merely became more evident with the greater stability of operating conditions. Since there was now no space available in the building for the addition of scavenger machines, experimentation in both the plant and the laboratory continued with a view to improving recoveries by less costly

TABLE II.— A REPRESENTATIVE MONTHLY METALLURGICAL SUMMARY

	Tons	Assays and Recoveries			
		Gold oz./ton	Silver oz./ton	Zinc %	Lead %
PRIMARY FLOTATION —					
Heads (classifier overflow)	30,000	0.037	4.28	3.82	0.50
Primary conc. (to cyanide circuit)	6,383	0.1543	18.14	5.38	1.79
Primary tails (to zinc flotation)	23,617	0.0053	0.53	3.40	0.15
Recovery in concentrate		88.7%	90.2%	30.0%	76.2%
Ratio of concentration	4.70				
CYANIDE CIRCUIT —					
Heads (primary conc.)	6,383	0.1543	18.14		
Pregnant solution	19,500	0.045	2.42		
Barren solution	19,500	Tr	0.006		
Tails (solids) (to zinc flotation)	6,383	0.016	10.70		
Tails (soluble)	3,192	0.0016	0.116		
Recovery		89.1%	40.7%		
ZINC FLOTATION —					
Heads (Prim. tails plus cy. tails)	30,000	0.0076	2.68	3.82	0.50
Concentrate (to de-leading)	1,948	0.0277	29.07	55.15	6.05
Tails (to waste)	28,052	0.0062	0.85	0.25	0.11
Recovery in concentrate		23.7%	70.4%	93.7%	78.6%
Ratio of concentration	15.40				
LEAD FLOTATION —					
Heads (conc. from zinc flotation)	1,948	0.0277	29.07	55.15	6.05
Concentrate (lead conc.)	226	0.1138	210.34	16.71	48.64
Tail (zinc conc.)	1,722	0.0164	5.28	60.20	0.46
Recovery in lead concentrate		47.7%	83.9%	3.6%	93.3%
Ratio of concentration	8.63				
OVERALL MILL RECOVERY —					
In precipitate (by cyanidation)		79.1%	36.8%		
In zinc concentrate (net)*			4.4%	90.5%	
In lead concentrate		2.3%	37.0%		73.3%
Overall recovery		81.4%	78.2%	90.5%	73.3%

*Payment is received for silver in the zinc concentrate in excess of 2.0 oz./ton.

TABLE III.— SHOWING DISTRIBUTION OF REAGENTS

Reagent	How Fed	Point of Addition	Rate Lb./Ton of		Remarks
			Mill Feed	Circuit Feed	
PRIMARY FLOTATION					
Aerofloat 31	Undiluted	Ball mills	0.018	0.018	
Aerofloat 208	10% soln.	Ball mills	0.06	0.06	
Sod. et. xan.	10% soln.	Class. o'flow	0.036	0.036	
" " "	" "	Primary cells	0.041	0.041	Stage fed
Pine oil	Undiluted	Primary cells	0.016	0.016	Stage fed
ZINC FLOTATION					
Copper sulphate	Sat. soln.	1st zinc conditioner	0.55	0.55	Varies with zinc heads
		2nd zinc conditioner	0.55	0.55	Varies with zinc heads
Lime (hydrated)	Milk of lime	2nd zinc conditioner	0.45	0.45	
" "	" "	Zinc cleaners	0.25		To 2nd last stage
" "	" "	Conc. thickener	0.25		Settling aid
Sod. et. xan.	10% soln.	Zinc roughers	0.042	0.042	Stage fed
Aerofloat 208	10% soln.	2nd zinc conditioner	0.002	0.002	As required
" "	" "	Head of scavengers	0.002	0.002	As required
Pine oil	Undiluted	Zinc roughers	0.004	0.004	Stage fed
DE-LEADING ZINC CONCENTRATE					
Sodium cyanide	15% soln.	1st conditioner	0.196	2.6	
" "	" "	Head of roughers	0.015	0.2	
" "	" "	1st stage cleaners	0.007		
Zinc sulphate	10% soln.	1st clonditioner	0.15	2.0	
Lime (hydrated)	Milk of lime	Conc. thickener	0.10		Settling aid
CYANIDATION					
Lime (hydrated)	Milk of lime	De-watering thickener	0.40	1.9	Settling aid
Aero cyanide	Dry	1st agitator	2.0	9.4	
" "	"	2nd agitator	0.80	3.8	
Zinc dust	Dry	Precipitation	0.085	0.4	0.053 lb./oz. precipitated
Mercuric chloride	Dry	1st agitator	0.06	0.3	Used periodically
Alchem 918	In solution	Clarifier	0.002	0.011	0.0036 lb./ton solution

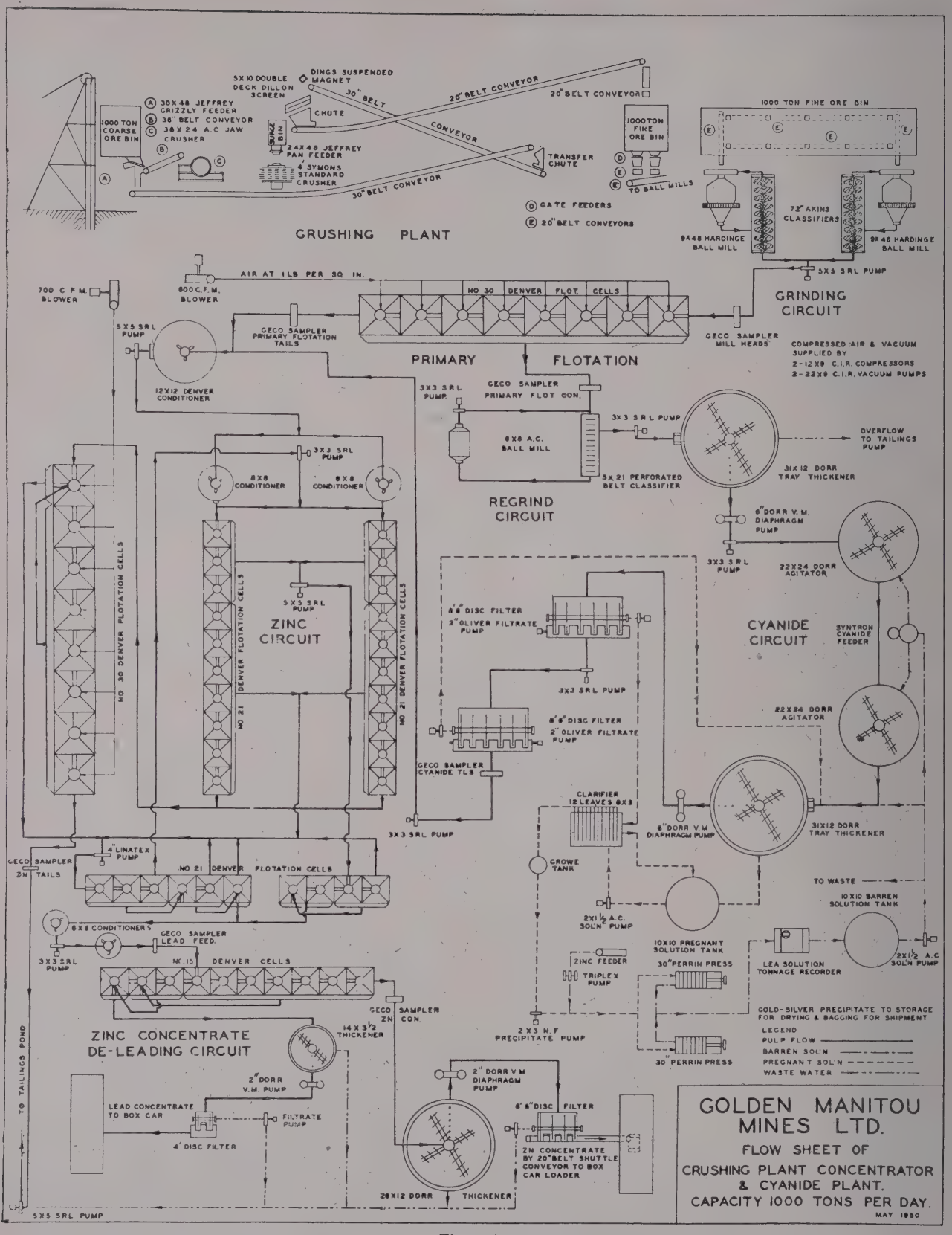

Figure 7.

means. Frequent manipulation of the zinc circuit, both mechanically and metallurgically, failed to produce the results desired. Losses were accentuated during periods of fluctuating mill-heads assays, with which the operators were unable to cope.

It was ultimately decided that an addition to the zinc circuit, involving an extension of the mill building, was the only feasible method of further improving recoveries. In order to handle the increased circulating load expected from the addition of scavengers, the zinc cleaning section and the zinc concentrate de-leading circuit following were also expanded. This equipment was put into service in January, 1950, and the improvement which resulted is illustrated by comparing the recovery figures at the bottom of Table II — representative of current operating results — with those of Table I.

In considering current metallurgical practice reference should be made to Figure 7, showing the present flow-sheet for the crushing plant, concentrator, and cyanide circuit. A metallurgical summary giving the distribution of the various metals will be found in Table II, and the reagent details are summarized in Table III.

Primary Flotation

Primary flotation involves the handling of 1,000 tons daily of solids with a specific gravity of 3.0, at a pulp dilution of 46 per cent solids, screening 60 per cent minus 200 mesh, to produce an uncleaned concentrate containing the highest possible percentage of the gold and the silver in the ore. Since appreciable amounts of precious metals are associated with the pyrite, it is aimed to maintain a high recovery of this mineral at this stage. Considering the fact that the pyrite content of the ore has varied from a low of 10 per cent to a high of 50 per cent, it is readily seen that the daily quantity of concentrate floated here, for the re-grinding and cyanidation circuits following, varies through a wide range. For reasons stated previously, no attempt is made either to depress or to activate sphalerite, with the result that the primary concentrate contains roughly 30 per cent of the zinc in the ore. As the pyrite content of the ore decreases, the zinc assay of the primary concentrate rises, but the percentage of the total zinc floated in it diminishes. The calculated contact time for this flotation circuit at full tonnage is 24 minutes.

Zinc Flotation

Zinc flotation involves the handling of 1,000 tons daily of combined primary flotation tailing and cyanidation tailing, in varying proportions, at a pulp dilution of 46 per cent solids in the conditioners, and 40 per cent solids in the rougher section. The contact times for conditioning, roughing, and scavenging, including a somewhat higher than average circulating load, are respectively 60, 16, and 20 minutes. The aim at this stage is to float as much zinc, lead, and silver as possible. A high-lime circuit is not considered favourable for the flotation of galena and the tennantite-tetrahedrite silver minerals. However, alkalinity in the roughers of 10.2 *pH* gives reasonably good recoveries of these minerals, with satisfactory pyrite depression. The relative floatability of the minerals in this circuit is shown in Figure 6.

The higher *pH* in the cleaners, which ranges from 11.5 to 12 plus, causes, at times, an appreciable rejection of lead and silver minerals and contributes to a high circulating load. Efforts to overcome, or at least minimize, this difficulty have resulted in a quite flexible and somewhat complicated flow in this section of the concentrator (Figure 7). The more common practice of re-circulating the concentrates from each roughing and scavenging step to the head of the preceding stage is not used. This, again, is due to the reluctance of the lead and silver to float in a high-lime pulp and therefore, once floated, every effort is made to route them directly to the cleaners, where they are introduced at a stage consistent with their zinc content. For example, the concentrate from the head end of the roughers, with its relatively high zinc content, goes directly to the second last or even the final stage of cleaning, while the lower grade scavenger concentrate, carrying quite appreciable quantities of lead and silver, more often than not enters the first cleaners, where, if desired, it can be subjected to four successive stages of cleaning. It has been established that, in this manner, a major portion of the circulating load of lead and silver minerals is confined to the cleaner circuit, thereby minimizing the tailing losses. The manipulation of this circuit is left to the discretion of the experienced operator, who is guided by assays at key points for zinc and lead every four hours. A recently installed six-point *pH* recorder facilities alkalinity control.

De-Leading of Zinc Concentrates

The zinc concentrate at this stage contains approximately 75 per cent of the lead and 45 per cent of the silver. This silver is not readily amenable to cyanidation. The problem of recovering these last two metals by flotation in a marketable lead concentrate, together with a small amount of copper — for which payment is also received — has already been mentioned and is covered in detail in the paper on de-leading of zinc concentrate referred to on an earlier page.

Current practice provides a conditioning period which averages about 120 minutes at 50 per cent solids, roughing capacity of 21 minutes at 30 to 35 per cent solids, including circulating load, and one or two stages of cleaning as desired. The zinc content of this concentrate is quite high and at times reaches 20 per cent. This situation can be improved by more thorough cleaning at the expense of lower silver recovery. It therefore becomes a matter of regulating conditions to yield optimum returns. The use of sodium sulphite in addition to the current reagents was previously believed to be beneficial, but was discontinued some time ago without noticeable loss in efficiency. The reagents used in this circuit are listed in Table III.

Cyanidation

Due to the fact that the recovery of silver has presented a major difficulty, extraction by cyanidation has received a great deal of attention in the testing laboratory. Space does not permit discussion of the many hundreds of tests which have been carried out in connection with this problem. This investigative work, conducted at the property and elsewhere, has revealed no advantage to be gained by a deviation from the simple conventional flow-sheet shown in Figure 7.

The product cyanided consists of primary flotation concentrate reground to approximately 90 per cent minus 200 mesh and thickened from 25 per cent to 67 per cent solids. This fineness has been found to give optimum results.

Over the years frequent changes in ore characteristics have resulted in a feed of varying composition, and the cyanide solutions have varied accordingly. Typical analyses given in Table IV show the more significant constituents of these products.

As pointed out under the heading of *Primary Flotation,* the daily tonnage to cyanidation varies with the pyrite content of the ore. Currently, the contact time in the agitators is about 40 hours, but in the past it has dropped to as little as half this time when the pyrite is high. The dissolution of gold is rapid — 90 per cent of the total extraction takes place in 3 hours, with very little improvement after 20 hours. The silver dissolution, on the other hand, is relatively rapid for the first 10 hours, gradually becomes slower during the next 30 hours, and then continues at a constant, but much reduced, rate for the next 40 hours, beyond which time it has not been investigated. Prolonged agitation, however, results in excessive cyanide consumption, exceeding the value of the additional silver extracted.

A cyanide strength approaching 5.0, expressed as pounds of KCN per ton of solution, as determined by the silver nitrate-potassium iodide titration, has been established as the optimum under average ore conditions

TABLE IV.— SHOWING ANALYSES OF CYANIDATION FEED AND PREGNANT SOLUTION

CONSTITUENT	CYANIDATION FEED	PREGNANT SOLUTION
Gold	0.12 oz./ton	0.039 oz./ton
Silver	9.28 "	1.39 "
Copper	0.50 per cent	0.0437 per cent
Lead	2.00 "	0.0006 "
Zinc	5.50 "	0.0205 "
Iron	32.23 "	0.0017 "
Arsenic	0.18 "	
Antimony	0.01 "	
Sulphur	40.20 "	
Insoluble	18.60 "	
Ferrous Iron	0.81 "	
Sodium thiocyanate		0.170 per cent
Soluble sulphide		Nil
Ferrocyanide		Trace
Sodium cyanide (titration)		0.129 "
Sodium cyanide (distillation)		0.235 "
Lime, CaO		0.032 "

(Figure 8). Both fineness and contact time are governed by silver requirements, since gold extraction is satisfactory with a much lower solution strength and a coarser grind. Regular laboratory control is employed to adjust plant conditions to variations in tonnage, silver head assays, and other relevant factors. The lime content of the solutions appears to have no bearing on extraction or cyanide consumption, and therefore only sufficient is maintained for thickening purposes.

Mercuric chloride as an aid to silver extraction, though not continuously employed, is used on occasions to advantage. The effect of this reagent — uncommon in cyanidation — is not fully understood. It is known that mercury salts help the dissolution of silver in the presence of double cyanides of zinc and copper, which have a poor solvent effect on silver sulphides. It is theorized, therefore, that mercuric chloride, in the presence of double cyanides, forms mercuric cyanide, which is soluble and an active solvent for silver sulphides. The benefit was especially marked prior to the addition of scavengers to the zinc flotation circuit, which have increased the silver recovery to the point where the use of mercuric chloride in cyanidation is not always economical, particularly in view of the fluctuation in cost between $1.00 and $4.00 a pound.

The reducing power of the cyanide solutions, as determined by the customary N/10 permanganate titration, reaches a high of approximately 1,000 at the end of agitation. Efficiency in the cyanide circuit does not seem to be affected by this factor, since the reducing power has been as high as 2,000 without apparent harmful effect on either extraction or precipitation. Generally speaking, no serious metallurgical problems have been encountered in precipitation. The quantity of barren solution wasted is governed, of course, by the intake of fresh water, which of necessity is quite heavy, chiefly due to the washing requirements of two stages of filtration ahead of the zinc circuit. An average solution balance is shown in Figure 9.

The use of disc filters for washing cyanide residue is a misapplication. The choice of this equipment was made owing to the uncertainty of the ultimate scheme of ore treatment. Unfortunately, it is impractical to substitute drum-type filters, which should give superior performance in reducing the soluble loss of precious metals, and at the same time minimize the residual cyanide going forward to zinc flotation.

General

Three products, *i.e.*, (1) precipitate from cyanidation, (2) zinc concentrate, and (3) lead concentrate, conforming to typical analyses shown

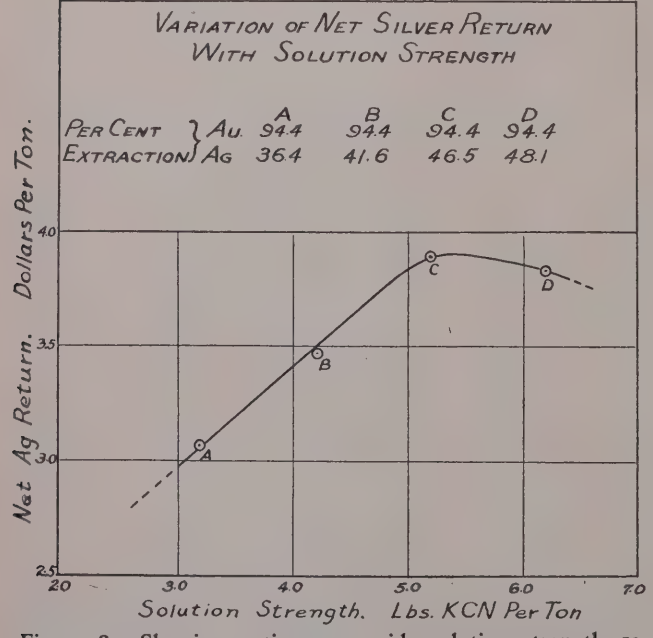

Figure 8.—Showing optimum cyanide solution strength as determined by net dollar return for silver.

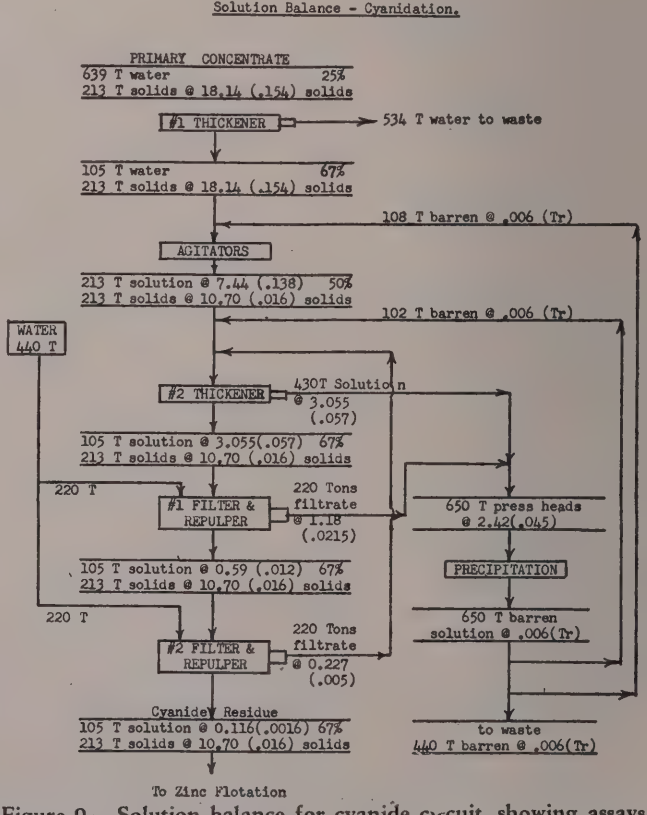
Solution Balance - Cyanidation.
PRIMARY CONCENTRATE
639 T water 25%
213 T solids @ 18.14 (.154) solids
#1 THICKENER
534 T water to waste
105 T water 67%
213 T solids @ 18.14 (.154) solids
108 T barren @ .006 (Tr)
AGITATORS
213 T solution @ 7.44 (.138) 50%
213 T solids @ 10.70 (.016) solids
102 T barren @ .006 (Tr)
WATER 440 T
#2 THICKENER
430T Solution @ 3.055 (.057)
105 T solution @ 3.055(.057) 67%
213 T solids @ 10.70 (.016) solids
220 T
#1 FILTER & REPULPER
220 Tons filtrate @ 1.18 (.0215)
650 T press heads @ 2.42(.045)
105 T solution @ 0.59 (.012) 67%
213 T solids @ 10.70 (.016) solids
PRECIPITATION
220 T
650 T barren solution @ .006(Tr)
#2 FILTER & REPULPER
220 Tons filtrate @ 0.227 (.005)
Cyanide Residue
105 T solution @ 0.116(.0016) 67%
213 T solids @ 10.70 (.016) solids
to waste
440 T barren @ .006(Tr)
To Zinc Flotation

Figure 9.—Solution balance for cyanide circuit, showing assays for gold and silver in oz. per ton (gold in parenthesis).

in Table V, are shipped respectively to the American Metal Company, New Jersey, the American Zinc Company, Illinois, and the American Smelting & Refining Company, Montana. The respective moisture contents of these products as shipped are approximately 0.4, 9.5, and 8.5 per cent. The precipitate coming from the presses carries 35 per cent moisture. It is dried to 0.4 per cent by means of infra-red lights supplemented by steam coils, sealed in 6-ply paper bags, boxed, and shipped monthly. Analyses of zinc concentrates from time to time have shown, in addition to the tabulated constituents, the following elements (per cent): cadmium, 0.22; antimony, 0.01; bismuth, 0.002; tin, 0.008; and a trace of cobalt.

Wide seasonal fluctuations in feed-water temperature and characteristics are variables over which there is no control. The water supply is obtained from a spring-fed creek of relatively small volume and therefore the run-off from the surrounding country during spring break-up and rainy periods affects its make-up appreciably. Generally speaking, it has been observed that zinc flotation improves in the late spring and summer months. The increased floatability of zinc has an adverse effect on the depression of sphalerite in the de-leading circuit to the point where some sacrifice in lead and silver recoveries is necessary to maintain acceptable lead concentrate grade. Laboratory tests have failed to confirm the effects noted and, as an adequate supply of steam has not been available for a plant test, it cannot be said with certainty whether temperature changes or variable water constituents, or both, account for the change in zinc floatability. It has, however, been found that, with the recent increase in flotation cell capacity, seasonal variations have had less effect on mill performance.

TABLE V.— SHOWING TYPICAL ANALYSES OF END PRODUCTS ON A DRY BASIS

CONSTITUENT	PRECIPITATE	ZINC CONCENTRATE	LEAD CONCENTRATE
Gold	423 oz./ton	0.016 oz./ton	0.114 oz./ton
Silver	21,290 "	5.28 "	210.34 "
Zinc	7.35 per cent	60.20 per cent	16.70 per cent
Lead	6.28 "	0.46 "	48.65 "
Copper	7.60 "	0.25 "	5.50 "
Iron	0.26 "	2.50 "	5.50 "
Arsenic	0.01 "	0.03 "	0.90 "
Sulphur	2.43 "	32.65 "	21.50 "
Insoluble (SiO_2. Al_2O_3)	1.46 "	3.67 "	0.70 "

As noted previously, the ore contains a variable amount of pyrite. In 1946 it reached a high of 50 per cent but has since diminished to a figure currently approximating 12 per cent. The production of a pyrite concentrate has been contemplated ever since before the commencement of operations, and it has been established that the only feasible place for its flotation is from the mill tailing. The addition of the scavenger flotation cells has reduced to a satisfactory level impurities such as zinc, lead, and arsenic, which formerly would have rendered the marketing of pyrite problematical. A deterrent which still exists, however, is the severe depressing action of the long contact time with cyanide and lime during cyanidation, which renders the pyrite less floatable than is ordinarily the case, and therefore more costly to produce. The current sulphur shortage has altered the picture appreciably and, as a consequence, pyrite investigations are again in progress.

ACKNOWLEDGMENTS

The writer wishes to thank the Management and Directors of Golden Manitou Mines for permission to publish this paper. In particular, he would thank Mr. Andrew Robertson, now Vice-President and General Manager, whose encouragement and active interest in the metallurgical developments have been most helpful. Grateful acknowledgment is made to Messrs. J. D. Wright and J. P. Jolicoeur, of the Mill Staff, for their assistance in compiling the data and preparing the manuscript. Appreciation is also expressed for laboratory work and technical information provided, from time to time, by the Denver Equipment Company (Canada), Limited, Imperial Chemical Industries (England), North American Cyanamid, Limited, the Bureau of Mines (Ottawa), and the Quebec Department of Mines.

Determination of Uranium in Ores

(*Transactions, Volume LV*, 1952, pp. 26-38.)

Review of Chemical Methods

By F. T. RABBITTS †

(*Annual General Meeting, Toronto, Ont. April*, 1950)

Introduction

WHEN the investigation of radioactive ores began at the Mines Branch about five years ago it was at once evident that improved chemical methods for the determination of uranium were essential. The published methods available were time-consuming and insensitive and were lacking in accuracy and precision. It was desirable to obtain methods which would be rapid, sensitive, and accurate. They should also allow for the possible presence of any interfering elements and should be adaptable to routine analysis by technicians.

Chemical methods are necessary to provide standard analysed samples for counting and to check analyses by counter methods. Moreover, counter methods cannot be used, and only chemical methods are suitable, when the sample is not in equilibrium, as with weathered minerals and leach residues. As leaching processes are improved, more sensitive chemical methods are required for the increasingly lower content of uranium in the leached residues. When the leaching process reaches the plant stage, speed of analysis becomes more important, without sacrificing accuracy.

The purpose of this paper is to describe briefly the methods used at the Mines Branch for the chemical analysis of uranium in ores. These methods represent the results of investigations by a number of groups in the United Kingdom, the United States, and Canada. They have been modified, where necessary, to be applicable to Canadian ores, many of which are complex.

Four methods are outlined. *Method 1*, the typical textbook method, is of historical interest only. *Method 2*, the *Mercury Cathode-Cupferron Method*, was formerly used in the Mines Branch but has now been replaced by *Methods 3 and 4*, the *Cellulose Column* and *Microfluorimetric* methods.

Methods

Method 1.—Typical Textbook Method

This method (Figure 1) is very long and the accuracy and precision obtainable are poor as it is difficult to ignite ammonium diuranate to a residue of constant composition.

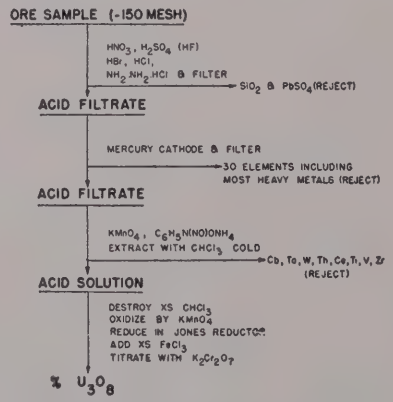

Figure 2.—Mercury cathode-cupferron method of analysis.

If vanadium is present, an additional step is required in which the vanadium is separated as lead vanadate.

No special apparatus is required.

Method 2.—Mercury Cathode-Cupferron Method

The detailed procedure for this method of uranium analysis has already been published in Mine Branch *Memorandum Series No. 103* (1). It is a general method for any type of uranium ore but is time consuming (see Figure 2).

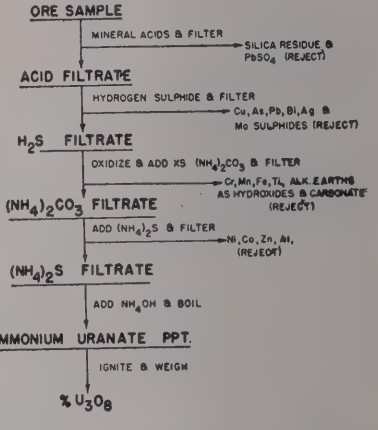
ORE SAMPLE
MINERAL ACIDS & FILTER
SILICA RESIDUE & $PbSO_4$ (REJECT)
ACID FILTRATE
HYDROGEN SULPHIDE & FILTER
Cu, As, Pb, Bi, Ag & Mo SULPHIDES (REJECT)
H_2S FILTRATE
OXIDIZE & ADD XS $(NH_4)_2CO_3$ & FILTER
Cr, Mn, Fe, Ti, ALK. EARTHS AS HYDROXIDES & CARBONATE (REJECT)
$(NH_4)_2CO_3$ FILTRATE
ADD $(NH_4)_2S$ & FILTER
Ni, Co, Zn, Al, (REJECT)
$(NH_4)_2S$ FILTRATE
ADD NH_4OH & BOIL
AMMONIUM URANATE PPT.
IGNITE & WEIGH
% U_3O_8

Figure 1.—Typical textbook method of analysis.

The accuracy and precision are good for ores containing 0.1 per cent U_3O_8 or better but colorimetric finish is not suitable because of the high concentration of salts present near the end of the analysis.

The only special equipment required is a set of mercury cathode cells. The type of cell used at the Mines Branch is illustrated in Figure 3.

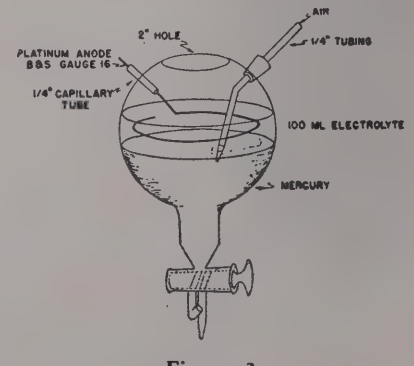

Figure 3.

*Published by permission of the Director-General of Scientific Services, Department of Mines and Technical Surveys, Ottawa.

†Radioactivity Division, Mines Branch, Ottawa.

(1) For references, see end of paper.

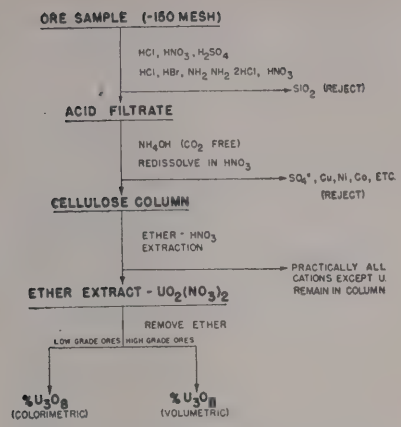

Figure 4.—Cellulose column method of analysis.

Method 3.—Cellulose Column Method

A full description of this method is given in the paper that follows. It is based on the selective extraction of uranyl nitrate by ethyl ether, the other cations being absorbed by the cellulose (see Figure 4). The procedure used at the Mines Branch is a modification of that developed by the Chemical Research Laboratory, Teddington, England (5).

The procedure given is a general one for complex ores, but can be shortened for less complex ores. For example, in some ores the uranium minerals are readily soluble in nitric acid. This acid alone could then be used as the solvent and the ammonia precipitation step to remove the bulk of the sulphate ions would thus be unnecessary.

The accuracy and precision of this method are good for ores containing 0.01 per cent U_3O_8 or better. For ores in the 0.01 to 0.05 per cent U_3O_8 range, the average mean deviation is approximately plus or minus 8 per cent, while for increasingly higher grade material the average mean deviation decreases steadily.

Eight determinations per day would be about average for one trained technician.

The method is readily adaptable for routine work and is recommended as the most suitable general method, since any degree of concentration of uranium can be extracted in the cellulose column.

It should be emphasized that ventilation and fire precautions must be adequate when handling large amounts of ethyl ether.

The only special glass apparatus required is the column illustrated in Figure 5.

For ores below 1 per cent U_3O_8, a colorimetric determination is more suitable. A good spectrophotometer such as the Beckman *B* or *DU* model is required. Most filter-type photometers are unsuitable for this determination.

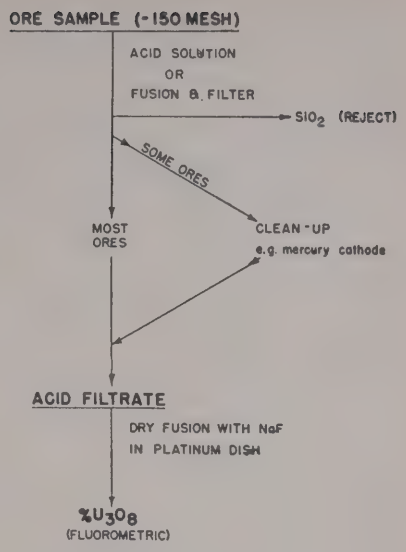

Figure 6.—Microfluorimetric method of analysis.

Method 4.—Microfluorimetric Method

Much of the development work on this method (see Figure 6) was done at the Argonne National Laboratory, Chicago, by G. R. Price, R. J. Ferretti, and S. Schwartz. Two of their published papers are *AECD 2282* and *AECD 2870* (4). The following

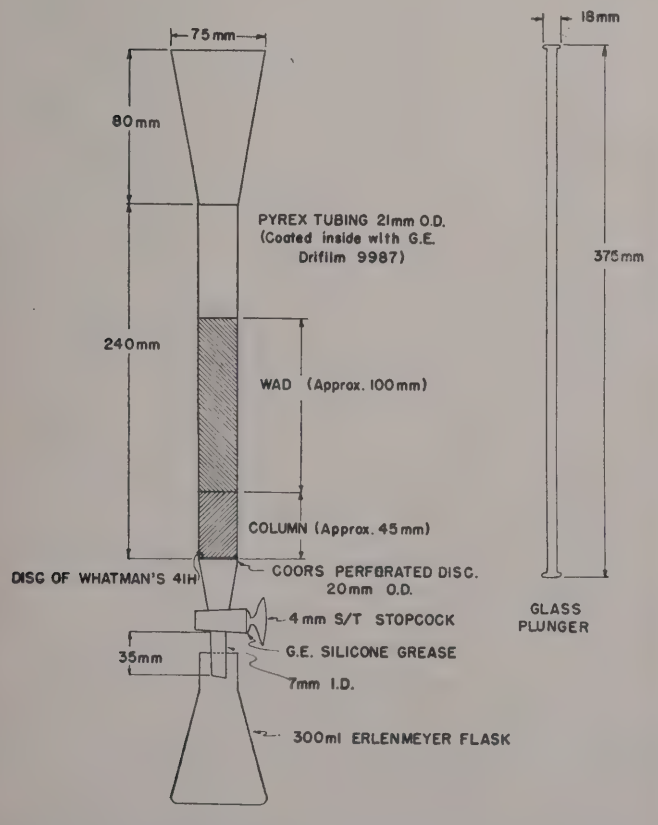

Figure 5.—Cellulose column for extraction of uranyl nitrate.

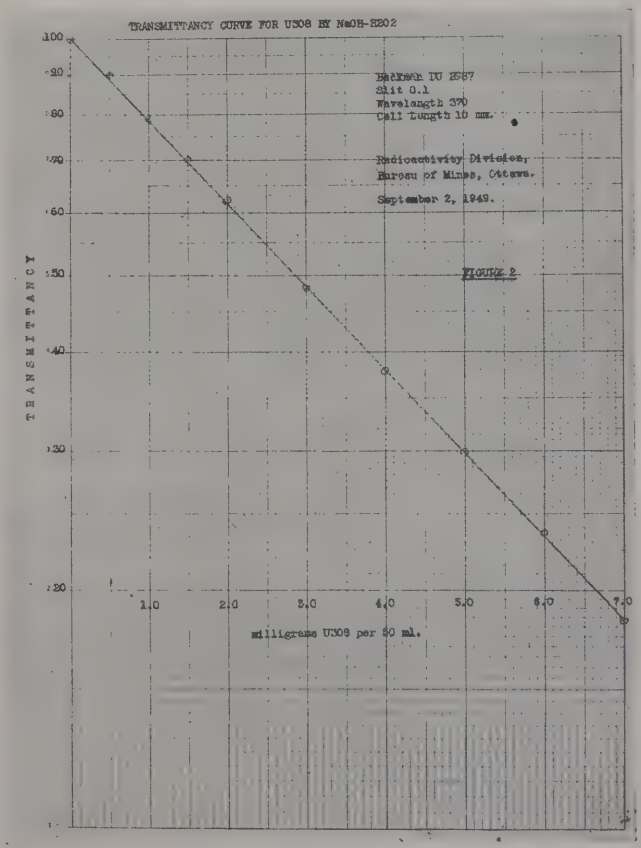

Figure 7.—Transmittancy curve for U_3O_8 by $NaOH$-H_2O_2.

abstract of Report *AECD 2870* is reproduced from *Nuclear Science Abstracts*, Vol. 4, No. 16, Aug. 30, 1950:

"The brilliant yellow-green fluorescence of uranyl salts provides a means for uranium determination which offers advantages possessed by few other analytical methods for any element.

"From an unknown solution, an aliquot of 0.005 to 0.1 ml. is pipetted into a shallow platinum dish about 1.5 cm. in diameter, and evaporated to dryness under a heat lamp. If a large residue of readily volatilizable or decomposable material remains, the sample in the dish is heated in a flame; only rarely is any other type of purification employed. To the dry residue in the dish approximately 0.3 gm. of sodium fluoride or sodium fluoride-sodium carbonate mixture is added, and the salt is fused by holding the dish in the flame of a Meker burner or by mounting it on a gas stove. After cooling, the disc of fused salt, either still in the platinum dish or 'tapped out' from the dish, is irradiated with light from the 365 millimicron mercury line and the yellow-green fluorescence is measured by a photoelectric fluorophotometer.

"From the reading obtained for the fluorescence of the sample, the average value for the fluorescence of blanks is subtracted, and the difference is multiplied by a previously determined calibration factor to give the amount of uranium in the sample.

"If there is reason, because of the presence of colour in the fused flux or from knowledge of the composition of the solution being analyzed, to suspect interference with the fluorescence, one can to a considerable degree remove this quenching by dilution, by employing a smaller aliquot, or can correct for it by 'spiking' by adding a known amount of uranium to an identical aliquot in another dish.

"The useful range of the method is for a sample size of from about 10 to 0.0001 micrograms of uranium; work below 0.001 micrograms requires great care in avoiding contamination. Usually, analyses are run in duplicate; for routine work the standard error of the mean of the pair is generally below 10 per cent except when the sensitivity is near its limit or when unusually large amounts of interfering substances are present, and by employing certain refinements in technique it is frequently possible to reduce the standard error to less than 5 per cent.

"No element besides uranium has been shown to produce detectable fluorescence under the conditions recommended in this paper".

This is a general method for ores containing 1 per cent U_3O_8 or less. The accuracy and precision obtainable are very good, especially for low-grade ores. At the Mines Branch, the average mean deviation for ores in the range 0.001 to 1.0 per cent U_3O_8 is approximately plus or minus 5 per cent. The method is rapid and is well suited to routine work but is not suitable for ores containing over 1 per cent U_3O_8. For this reason, it is best used in conjunction with the cellulose column method.

The capital outlay for this method of uranium analysis is comparatively high. A suitable fluorimeter will cost between $1,500 and $2,000, while the investment in platinum dishes will range from $500 to $1,000. In addition, it is desirable to operate the fluorimeter in an air-conditioned room since it is important to control atmospheric dust and humidity carefully. On the other hand, the cost of analysis per sample is very low once the equipment has been set up.

Using this method, an average output of ten ore samples or forty determinations can be obtained in an eight-hour day by a staff of two chemists.

References

(1) *The Determination of Uranium in Ores: Modified Mercury Cathode-Cupferron Method;* Memorandum Series No. 103, Mines Branch, Ottawa, August, 1949.

(2) *The Determination of Uranium in Ores and Solutions: Cellulose Column Method;* Memorandum Series No. 105, Mines Branch, Ottawa, September, 1949.

(3) *The Microfluorimetric Determination of Uranium;* United States Atomic Energy Commission Report No. AECD-2282.

(4) *The Fluorophotometric Determination of Uranium;* United States Atomic Energy Commission Report No. AECD-2870.

(5) *Handbook of Chemical Methods for the Determination of Uranium in Minerals and Ores;* H. M. Stationery Office, London, 1950.

Appendix

Reporting of Uranium or Thorium Determinations

Under the Atomic Energy Regulations of Canada the results of an assay or analysis of a mineral that indicates a content of more than 0.05 per cent by weight of uranium or thorium are to be reported forthwith to the Director of the Geological Survey, Department of Mines and Technical Surveys, Ottawa, for the Atomic Energy Control Board, together with full particulars relating to the material assayed or analysed, including the name and address of the person from whom such material was received, the purpose of the assay or analysis, and the origin of the material so far as known to the person making the report. This requirement does not apply to assays or analyses made for persons operating under orders of the Board which provide for periodical reports.

Copies of the Regulations may be obtained on application to the Secretary, Atomic Energy Control Board, Ottawa.

Laboratory Tests of the Cleaning of Fine Coal by a D.S.M. Cyclone

By T.E. MORIMOTO *

(*Annual Western Meeting, Calgary, Alta., October,* 1951)

(*Transactions, Volume LV,* 1952, *pp.* 39-47).

SUMMARY

THE Research Council of Alberta has made a series of tests with the cyclone washer to investigate the influence of the following operating variables: underflow apex diameter, specific gravity of medium, pulp ratio, density distribution of the feed, and size analysis of the feed. The results obtained for all the variables studied, except size analysis of the feed, are included in this paper. The tests were made with smaller underflow apex diameters and lower specific gravity suspensions than have been recommended by Cerchar for the sharpest separations. However, what is considered to be sufficiently good cleaning was accomplished with a suspension of 1.30 specific gravity and an underflow to overflow diameter ratio of 0.56. It was found that the sharpness of separation was not affected appreciably by variations in specific gravity of suspension or underflow apex diameter provided that the density of separation was within the normal range of washery operation.

Cyclone performance was not too adversely affected by decreasing the pulp ratio. The minimum figure which gave reasonable separation was 4 to 1.

The tests with various density distributions of feed revealed that the degree of washing difficulty had no detrimental effect on the sharpness of separation given by the cyclone.

INTRODUCTION

A major problem of the bituminous coal industry today is that of reducing the ash content of the finer sizes (minus ¼-inch) of coal. This problem is particularly acute in Alberta, where, because of tectonic movement subsequent to deposition, the coals have been rendered friable, and bands of impurities have been intimately mixed with the clean material. As a consequence, most Alberta coals, as mined, contain a large proportion of fines and have a high inherent mineral-matter content. The advent of mechanized mining and stripping operations has further aggravated this problem.

The cleaning of coal larger than ¼-inch is not too difficult. Jigs and heavy media washers in most cases give reasonably efficient separations if the coal is properly sized before treatment. The cleaning of fine coal, however, is a much more formidable task. Jigs and heavy media processes are inefficient when used on fine coal, and dry pneumatic equipment generally has proven to be unsatisfactory, particularly when the feed is damp. A recently developed vacuum jig (8) is reported to give good cleaning of fine coals when operating under satisfactory conditions, but no operating data are available. In some cases, wet tables give good results but have the disadvantage of requiring large floor space. Froth flotation appears at present to be the only method of satisfactorily cleaning coal smaller than ¼ mm. (approximately 60 mesh). However, it has the disadvantage of high cost.

The Research Council of Alberta, in undertaking research on the cleaning of fine coal, decided that equipment which used centrifugal force to aid in the cleaning action offered promise of extending the high efficiency of heavy media cleaning to the finer sizes of coal. Such a piece of equipment, the D.S.M. cyclone washer, had been developed by Dr. M. G. Driessen while working in the Dutch State Mines Laboratories at Limburg, Holland.

Several reports on the cyclone cleaner have been published, notably those of Driessen (4), Geer and Yancey (5), and more recently Belugou and de Chawlowski (2) and Burton (3). However, it was felt that more fundamental data were needed on the operating characteristics of the cyclone, particularly when cleaning coals with a high inherent mineral matter content, before any assessment could be made of its suitability for use in Alberta. Accordingly, the Council instituted work on the cleaning of fine coal with a three-inch-tured by the Dorr Company. This diameter D.S.M. cyclone manufacpaper reports on the progress that has been made to date.

EQUIPMENT AND TEST PROCEDURE

Figure 1 shows the construction of the D.S.M. cyclone. It consists of a conical section six inches high and three inches maximum diameter. Above the conical section is a cylindrical section called the feed chamber and directly above this part is the vortex finder and overflow chamber. The feed enters through a tapered inlet at the left, which is joined tangentially to the cylindrical feed chamber. A rubber ring, the diameter of which is adjusted by tightening or loosening an adjustable tail piece and a constriction ring, is used to regulate the refuse discharge at the bottom. Removable shims can be inserted to adjust the size of the feed inlet. However, these have not been used as it was desired to make all tests with the greatest throughput possible.

A photograph of the test set-up is shown in Figure 2. Batch tests only have been made to date, although it is hoped to make continuous runs at a future time. The cyclone is installed

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(8) For references, see end of paper.

over a sixty-gallon mixing tank (*a*) which is provided with baffles and a two-inch bottom outlet. A one-half horsepower Lightnin Mixer (*b*) gives efficient mixing of the slurry. A one-inch centrifugal sand pump (*c*) draws the feed slurry from the tank and delivers it to the feed inlet of the cyclone (*d*). Overflow and underflow from the cyclone are returned to the mixing tank. However, provision has been made for collecting and weighing timed samples of the overflow and underflow streams by simultaneously directing the flows into two weighing tanks. The trough (*f*) carries the underflow stream containing the refuse into the weighing tank (*g*) and the overflow stream containing the clean coal goes *via* the overflow pipe (*h*) into the weighing tank (*j*).

The procedure adopted for a coal cleaning test was as follows: The suspension of magnetite and water was brought to the required specific gravity and a quantity of fine coal (—10 +100 mesh) was added. The amount of coal added depended upon the desired pulp ratio*. The pump was then started, the valve to the cyclone feed inlet opened, and the inlet valve to the pump regulated until the required pressure was indicated on the cyclone feed inlet pressure gauge. When the operator was ready to collect samples of each stream, the flows were simultaneously directed into the two weighing tanks for a timed interval. At the end of this time, the streams were returned to the mixing tank.

The overflow slurry consisting of washed coal, magnetite, and water was then pumped through a belt-type laboratory size magnetic separator to recover the magnetic medium and to give a clean coal product. The washed coal was dried, weighed, and sampled for ash and float-sink analysis. After the washed coal slurry had been put through the magnetic separator, the refuse slurry was treated in the same manner to recover the magnetite medium and give a refuse

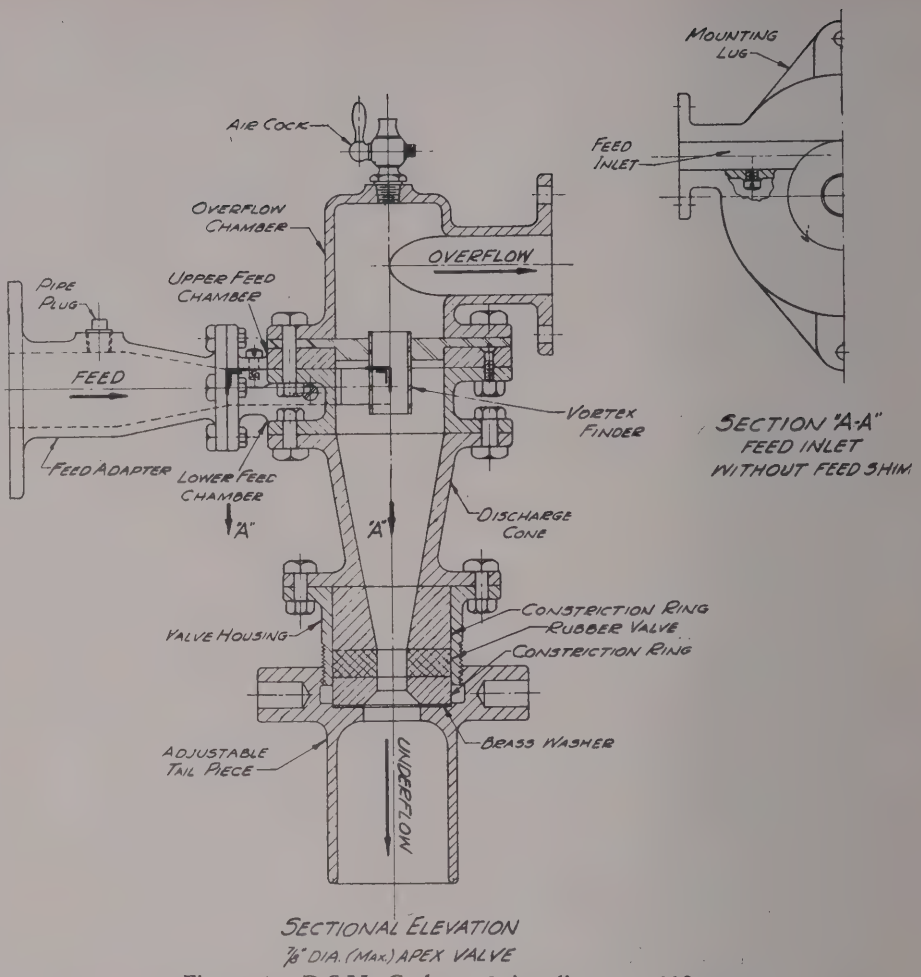

Figure 1.—D.S.M. Cyclone, 3 in. diameter, 20° cone.

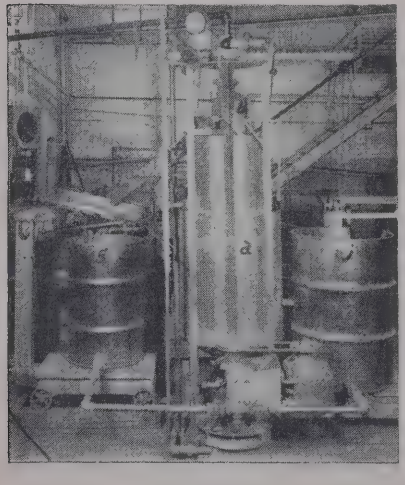

Figure 2*a*.—Experimental cyclone set-up for batch tests.

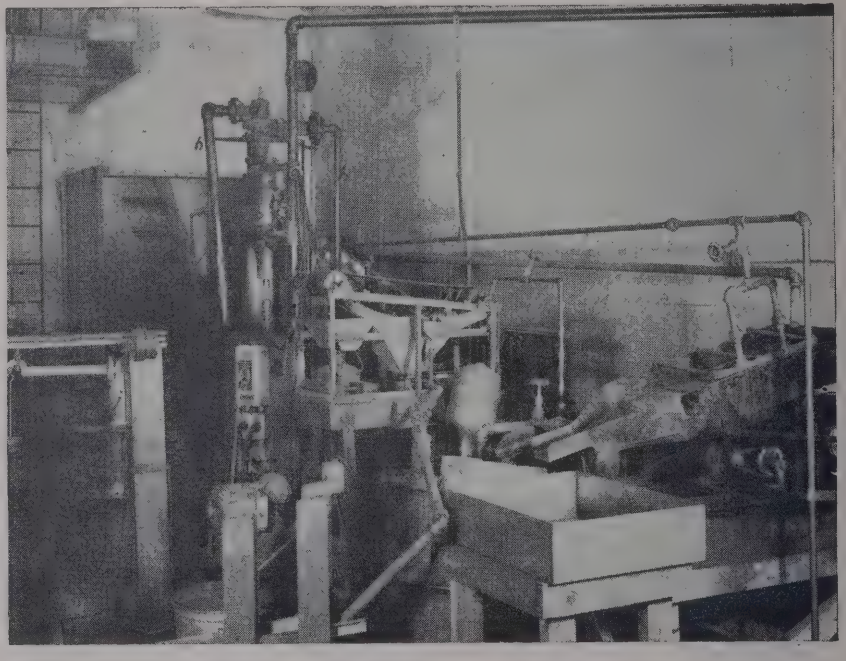

Figure 2*b*.—Another view of cyclone test set-up, showing magnetic separator and 100 mesh shaker screen.

*The term 'pulp ratio' is defined here as the ratio of the weight of magnetite and water suspension to the weight of coal. A pulp ratio of six to one was used for most of the tests in this report.

product which was dried and weighed for ash and float-sink analysis.

After several cleaning tests had been conducted it was found that considerable degradation of the coal occurred. This degradation made it very difficult to correlate the results, since the ash of the feed as calculated from the products varied by several per cent, depending on the length of time the coal was in the cricuit. The time of immersion of the washed coal and refuse in the circuit was excessive, because of the limited capacity of the small magnetic separator and the necessity of two passes for each product through the equipment. For this reason it was decided to forego magnetic separation. Therefore, for all tests in this investigation, the medium was removed from both washed coal and refuse products by spraying with water on a 100 mesh shaker screen. The immersion time was thus reduced from two or three hours to fifteen or twenty minutes. There was still considerable degradation, since the ash of the raw feed was approximately 19 per cent while the ash of the feed as calculated from the products was approximately 17 per cent. However, this calculated ash figure was fairly constant, ranging from 17.0 to 17.5 per cent.

Coal Used

The coal used in this investigation was a medium-volatile bituminous rank coal obtained through the courtesy of the Cadomin Coal Company, Ltd. It was crushed and screened to give a sized product between 10 and 100 mesh. Coarse material could not be used in a three-inch cyclone and 100 mesh was considered to be the finest size that could be recovered on the rinsing screens. This size range is greater than that used by Geer and Yancey (5) (—14+35 mesh) but it was decided to extend the lower limit to see whether or not cleaning could be accomplished on the finer sizes.

The same feed coal was used for most of the runs in order to keep the number of variables to a minimum. However, various types of feed were made up from this coal by blending with it a portion of the 1.40 to 1.80 specific gravity fraction or with the sinks heavier than 1.80. This gave the variable feed which is encountered in actual commercial cleaning plant operation.

Suspension

The medium used was a suspension of finely ground magnetite and water. The magnetite, designated *Magnetite B,* was obtained from the Foote Mineral Company, Philadelphia, Pa. The size specification given for the material was that 95 per cent passed through a 325 mesh screen. Magnetite was chosen for the initial test work because it was believed that magnetic separation of the magnetite from the coal might be the answer to the medium recovery problem. In addition, since magnetite has a high density (4.6 to 4.7), the amount needed to give the required specific gravity of suspension is small, and the medium viscosity is lower than would be the case for most other materials.

Theory

A thorough discussion of the theory of the cyclone washer is given by Driessen in his paper presented to Institute of Fuel (4). In it he states that the important factor in coal cleaning is the velocity with which particles rise or fall in a liquid. Upon examining Stokes Law and Allen's formula for settling, it becomes apparent that the four possible ways of increasing the settling velocity are (*i*) to increase the particle size, (*ii*) to increase the difference between the density of the particles and that of the liquid, (*iii*) to decrease the viscosity of the separating liquid, and lastly, (*iv*) to increase the acceleration of gravity. The first three methods are not feasible since particle size, and density of coal and separating liquid, are fixed, and the practical limit of viscosity is probably set by the viscosity of water. This leaves the last factor of increasing the acceleration of gravity, g. It can be shown that the settling velocity equations would remain unchanged except for g being replaced by u^2/r, where

u = circumferential velocity of the fluid and particles

r = radius of rotation

It would therefore be advantageous to increase the forces acting on the particles by centrifugal methods. Centrifuges, although they develop centrifugal forces several thousand times that due to gravity alone, are not used for coal cleaning. This is because they require a true liquid such as carbon tetrachloride or a suspension of particles which are so small that they would not settle if subjected to centrifugal force. A true liquid such as carbon tetrachloride has the disadvantage of high cost and the particle size of suspensions normally used (loess, barytes, magnetite, etc.) is so large that the particles would tend to separate out, leaving clear water instead of a suspension as the medium.

A device which uses centrifugal force to separate coal and shale but which overcomes the difficulties encountered by the centrifuge is the D.S.M. cyclone. The cyclone may be used as a thickener, in which case the working principles are much the same as in the well-known air cyclone. It can be used as a separator as well as a thickener; *e.g.,* a mixture of coal and shale can be separated, if a raw coal mixture in a heavy liquid such as carbon tetrachloride (specific gravity 1.58) is forced through the cyclone, the coal coming out at the top and the shale leaving through the apex at the bottom. The liquid may now be replaced by a suspension and a similar separation will occur. The suspension particles do not settle out as they do in the centrifuge; instead, a state of equilibrium seems to exist between the centrifugal forces acting on the particles and the forces due to the liquid flow, which act inwardly.

An explanation of the working principle of the cyclone which has been suggested by Driessen is as follows: "Raw coal is fed to the cyclone inlet, together with a suspension, the specific gravity of which can be considerably lower than the required separating gravity. Due to the thickening effect of the cyclone, a washing bed is formed in which the suspension particles are kept in equilibrium between the centrifugal forces and the dragging forces of the water current. The coal particles lighter than the effective specific gravity of the established washing bed are swept towards the outlet opening and the shale particles heavier than the specific gravity of the washing bed leave the cyclone at the apex".

Evaluation of the Washing Tests

In this investigation, it was necessary to determine the relative importance of the operating variables by conducting a series of tests in each of which there was, as far as could be ascertained, only one variable. The relative importance of any variable can then be appraised by the use of some reliable basis of comparison. There are several criteria that might be considered for the evaluation of coal washing apparatus. Among these are the Fraser-Yancey Efficiency (9), the Anderson Efficiency (1), Driessen's Error Area (4), the Probable Error developed by the French 6), and many others.

The most widely used of the above methods is probably the Fraser-Yancey Efficiency. It is the ratio, expressed in per cent, of the actual yield of washed coal to the yield of float coal of the same ash content shown to be present in the feed by a float-and-sink analysis. In washery operation, where the quality of washed coal and the yield are of prime

interest to the operator, this formula is quite useful, since it gives a direct measure of the loss of saleable coal.

However, the Fraser-Yancey Efficiency has several drawbacks which limit its usefulness as a criterion for assessing the results of this investigation. Firstly, this efficiency is a function not only of the performance of the washer, but also of the character of the coal being treated, the specific gravity at which the separation between coal and impurity is made, and the density distribution of the feed coal. Secondly, this formula has a serious shortcoming, in that, as the yield of washed coal approaches 100 per cent, the efficiency value also approaches 100 per cent, with the anomalous result that, if no refuse is removed, the efficiency for the separation is 100 per cent. Therefore, the Fraser-Yancey Efficiency could only be regarded as a criterion if all the tests were conducted on the same coal and all the separations were effected at the same specific gravity (9). A third factor which affected its use in evaluating the results of this investigation was degradation of the feed. Despite all precautions taken to limit this degradation, the ash of the feed as calculated from the products was approximately 17 per cent as compared with around 19 per cent for the raw feed coal. In a case such as this, the efficiency formula is not valid unless the cumulative float-ash curve is plotted from a calculated feed based on the products. If the cumulative float-ash curve is plotted from data given by analyses of the raw feed, values of more than 100 per cent efficiency are sometimes obtained. This is especially so when the separations are made at higher densities. An example illustrating this point is shown by Test 39 for which the efficiency as computed from the original raw feed curve is 102.5 per cent whereas that given by the calculated feed curve is 99.5 per cent.

An attempt was made in this investigation to use as perfectly uniform a feed as possible, but the separations were performed at various densities. Therefore, it was felt that the efficiency formula was not a suitable basis for comparing the performance of the cyclone in these tests except in those instances where the densities of separation were approximately equal. All Fraser-Yancey Efficiency values reported in this paper are based on a calculated feed.

A means of showing the sharpness of the separation effected by a coal washer, which is believed to be substantially independent of the character of the feed coal, is the Tromp distribution curve (7). This curve is derived from float-and-sink analysis of the products. The percentage of the feed reporting to washed coal is plotted versus density of the feed coal. The separation density at which the cut is made between washed coal and refuse is given by the density of the coal at the 50 per cent point. An example of a distribution curve is shown in Figure 3. Examination of this diagram shows that the slope of the curve is a measure of the sharpness of separation, *i.e.*, the steeper the slope, the sharper the separation. Theoretically perfect cleaning is illustrated in Figure 3 where the slope has become infinite, or, in other words, the curve has become a vertical line at the density of separation.

Several methods of expressing the efficiency of separation based upon the Tromp distribution curve have been developed, notably the Error Area by Driessen, the Anderson Efficiency, and the Probable Error which has been proposed by French investigators. The method of Probable Error seemed to be the easiest to compute and was believed to be substantially independent of the density of separation. This is a point, however, on which there is some disagreement. Vissac, Tromp, and Cerchar seem to be agreed that the probable error increases with higher separation densities. On the other hand, Yancey and Geer maintain that the probable error is virtually independent of the density of separation, at least for the range in which washeries are usually operated. They give data to substantiate this stand. The results obtained in this investigation would seem to verify those of Yancey and Geer, in that the probable error is not affected appreciably by the density of separation within the range of densities normally used.

Operating Variables Investigated

Variables which may influence the operation of a cyclone washer are: inlet feed opening, vortex finder diameter (overflow), inlet feed pressure, underflow apex diameter, specific gravity of the medium, pulp ratio of the slurry, size analysis of the feed, and density distribution of the feed coal. The effect of all excepting the first three variables was studied. Previously, Geer and Yancey had shown that inlet feed opening had little influence on cyclone performance. For this reason and because optimum throughput was desired, the maximum inlet feed area of 0.78 square inches was employed for this investigation. Further, it was considered that the feed inlet and overflow opening should have approximately the same area in order to give the greatest efficiency of operation, since most of the feed slurry goes out through the overflow. Therefore, a vortex finder diameter of 1.03 inches and area of 0.83 square inches was used for all tests. The maximum inlet feed pressure that could be maintained by the feed pump was 13.5 p.s.i.g. This pressure was used for all runs because the work of Geer and Yancey (5) had indicated that increased efficiency was given by higher pressures. The throughput of raw coal at this pressure and a pulp ratio of 6 to 1 was 1.2 tons per hour. The results obtained for all of the variables studied, except those of size analysis of the feed, are included in this paper.

Underflow Apex Diameter

The underflow apex diameters used in this investigation were considerably less than the vortex finder overflow diameter of 1.03 inches, the maximum being 0.82 inches and the minimum 0.58 inches. The French Cerchar investigations (2) revealed that the best cyclone operating results were given when the overflow and underflow exit diameters were approximately equal and the specific gravity of the medium was close to the density of separation. This means that, in order to carry out a separation at a density of 1.60, a suspension of the same order of specific gravity would be needed. It was felt that, in practice, it would probably be more economical to sacrifice a slight sharpness of separation for reduced costs of medium and medium recovery equipment by using a lower specific gravity suspension of 1.45 or less. In order that these lighter suspensions could be utilized, the ratios of underflow to overflow exit diameter had to be much less than the ratio of one to one recommended by Cerchar (2). The maximum ratio used was 0.79 and the minimum 0.56. By employing smaller underflow diameters, use can be made of one of the cyclone's advantages over other heavy media methods of cleaning, in that the separation is made at a much higher density than the specific gravity of the suspension used.

The probable errors quoted by Cerchar (2) for the cyclone washer lie between 0.020 and 0.045. The probable errors procured in these tests ranged from 0.040 to 0.092. However, it was believed that a probable error of no more than 0.075 was acceptable as long as the loss of clean coal was not excessive. This arbitrary figure

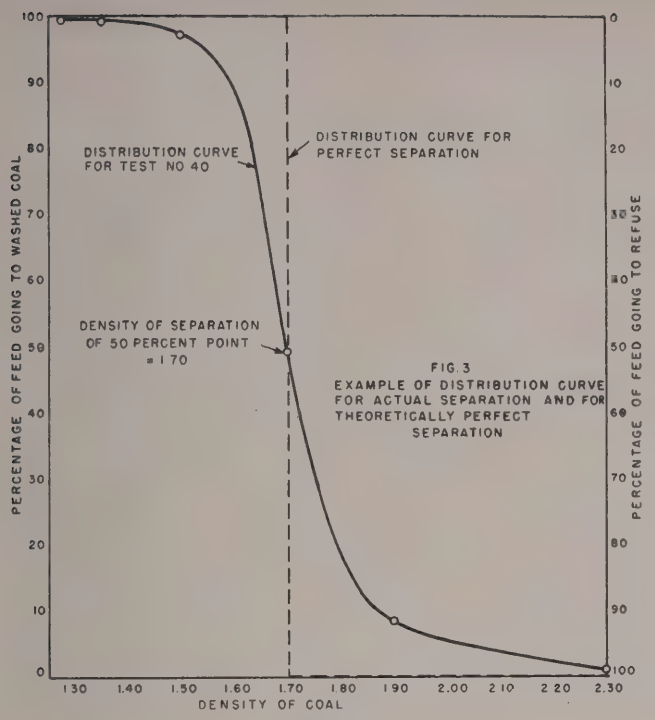
DISTRIBUTION CURVE FOR TEST NO 40
DISTRIBUTION CURVE FOR PERFECT SEPARATION
DENSITY OF SEPARATION OF 50 PERCENT POINT = 1.70
PERCENTAGE OF FEED GOING TO WASHED COAL
PERCENTAGE OF FEED GOING TO REFUSE
DENSITY OF COAL

FIG. 3
EXAMPLE OF DISTRIBUTION CURVE FOR ACTUAL SEPARATION AND FOR THEORETICALLY PERFECT SEPARATION

Figure 3.

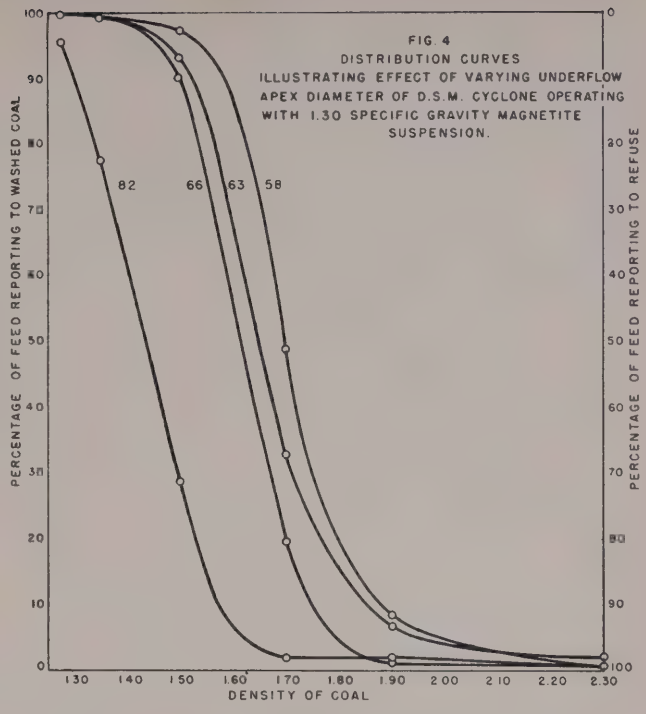

FIG. 4
DISTRIBUTION CURVES ILLUSTRATING EFFECT OF VARYING UNDERFLOW APEX DIAMETER OF D.S.M. CYCLONE OPERATING WITH 1.30 SPECIFIC GRAVITY MAGNETITE SUSPENSION.

Figure 4.

of 0.075 has been used as the maximum allowable probable error in evaluating the results of these tests.

Although the cyclone could have been equipped with a larger underflow apex diameter than the 0.82 inches employed, this was not done for the reason previously mentioned of keeping the suspension specific gravity as low as possible. Preliminary experiments determined that the minimum underflow apex diameter was 0.55 inches under the test conditions, since any opening smaller than this did not give a 'spray' discharge. When the underflow opening was made too small, a 'rope' type of discharge appeared. The difference between a 'rope' and a 'spray' discharge was easily discernible during operation of the cyclone. When a 'spray' discharge was being maintained, the sound of air rushing through the apex of the cone was clearly audible, the inlet pressure gauge remained steady, and the underflow discharge was in the form of a vortex which sprayed around the inner edge of the tailpiece. With a 'rope' discharge there was no rush of air, the pressure gauge was often unsteady, and the underflow discharge had the appearance of a rope. A rope discharge was undesirable since, with it, excessive underflow product was carried over into the overflow.

Influence of Underflow Apex Diameter

Table I shows the results of three series of tests. The specific gravity of the medium was kept at 1.30 for the first series and all other variables were held constant except the underflow apex diameter. The other series of tests were run with specific gravity suspensions of 1.40 and 1.45 respectively. The distribution curves for one of the three series of experimental runs at constant specific gravity of 1.30 are shown in Figure 4. A con-

Table I.— Influence of Underflow Apex Diameter on Cyclone Performance

	Specific Gravity of Medium									
	1.30				1.40			1.45		
Test No.	40	32	49	54	35	30	27	39	29	26
Underflow apex diam. in.	0.58	0.63	0.66	0.82	0.58	0.63	0.82	0.58	0.63	0.82
Yield of washed coal, %	81.0	82.0	79.7	55.0	88.5	88.0	78.5	91.2	90.0	82.2
Ash of washed coal, %	9.5	9.4	9.1	6.5	11.5	11.3	8.5	12.2	11.7	9.2
Ash of refuse, %	51.2	53.0	51.1	30.8	62.6	62.1	49.5	68.3	64.1	55.0
Density of separation.	1.70	1.65	1.61	1.43	1.81	1.78	1.56	1.98	1.84	1.65
Probable error	0.067	0.073	0.067	0.078	0.069	0.074	0.040	0.092	0.074	0.04
Fraser-Yancey* efficiency, %	97.4	97.2	97.5	77.0	97.7	98.4	97.2	99.5	99.2	97.5

The following conditions were constant for all the above tests with the 3-inch-diameter cyclone:

Feed coal	Cadomin −10+100 mesh
Pulp ratio	6 to 1
Inlet feed pressure	13.5 p.s.i.g.
Feed inlet	0.78 in. × 1.00 in.
Overflow vortex finder diameter	1.03 in.

*These Fraser-Yancey Efficiency values are slightly different from those given in the original manuscript. The above values were obtained from individual cumulative float-ash curves and are consequently more accurate than the original values, which were computed from an average cumulative float-ash curve.

sideration of the data in Table I discloses that the only probable errors over 0.075 were those for the largest overflow diameter of 0.82 inches with a 1.30 suspension and for the smallest underflow diameter of 0.58 inches with a medium of 1.45 specific gravity. These two tests were made at separation densities of 1.43 and 1.98, which are outside the normal range of washing operation. Therefore, if these two tests are disregarded, it is apparent that the resulting probable error values were fairly constant. The sharpest separations were represented by the two probable error values of 0.040 and 0.042. These latter separations at densities of 1.56 and 1.65 were accomplished with the largest underflow diameter of 0.82 inches using suspensions of 1.40 and 1.45 specific gravity respectively. These lower probable errors of 0.040 and 0.042 with the larger apex diameter tend to verify the findings of Cerchar that the cyclone performs most efficiently when the underflow and overflow exits are approximately equal and the specific gravity of suspensions approaches that of the separation density. It is likely that, if suspensions of higher specific gravity than 1.45 and larger underflow diameters than 0.82 inches had been used in this investigation, the probable error values resulting would have been lower than 0.040.

However, a comparison of one of the more efficient separations using a medium of 1.45 specific gravity and the largest underflow diameter of 0.82 inches with a separation using a lighter suspension of 1.30 specific gravity and a small apex diameter of 0.63 inches reveals some interesting facts. *e.g.*, compare the data for tests 26 and 32. Test 26 gave the far sharper separation with a probable error of 0.042, as compared with 0.073 for test 32. The yields were almost identical, 82.2 and 82.0 per cent respectively. The ash content of the washed coals was 9.2 and 9.4 per cent, while the ash content of the refuse was 55.0 and 53.0 per cent respectively. The calculated ash of the feed was 17.3 and 17.4 per cent respectively. The Yancey efficiencies of 97.5 and 97.2 per cent can be compared in this case, since the density of separation was 1.65 for both tests. Thus it is apparent that test 26 gave slightly the better performance in all respects. However, from the viewpoint of the coal operator, there may not be too much to choose between the two runs, since the yields are approximately the same and the ash contents of the products do not differ greatly. The economics of using a lighter suspension of 1.30, rather than one of 1.45, specific gravity would probably dictate the use of the former and therefore the smaller apex diameter.

TABLE II.— INFLUENCE OF SPECIFIC GRAVITY OF SUSPENSION ON CYCLONE PERFORMANCE

	Underflow Apex Diameter (inches)								
	0.82			0.63			0.58		
Test No.	54	27	26	32	30	29	40	35	39
Specific gravity of suspension	1.30	1.40	1.45	1.30	1.40	1.45	1.30	1.40	1.45
Yield of washed coal, %	55.0	78.5	82.2	82.0	88.0	90.0	81.0	88.5	91.2
Ash of washed coal, %	6.5	8.5	9.2	9.4	11.3	11.7	9.5	11.5	12.2
Ash of refuse, %	30.8	49.5	55.0	53.0	62.1	64.1	51.2	62.6	68.3
Density of separation	1.43	1.56	1.65	1.65	1.78	1.84	1.70	1.81	1.98
Probable error	0.078	0.040	0.042	0.073	0.074	0.074	0.067	0.069	0.092
Fraser-Yancey* efficiency, %	77.0	97.2	97.5	97.2	98.4	99.2	97.4	97.7	99.5

*See footnote to Table I.

The following conditions were constant for all the above tests with the three-inch-diameter cyclone:

Feed coal	Cadomin −10+100 mesh
Pulp ratio	6 to 1
Inlet feed pressure	13.5 p.s.i.g.
Feed inlet	0.78 in. × 1.00 in.
Overflow vortex finder diameter	1.03 in.

A possible explanation for the probable error for test 26 being so much less than that for test 32 (0.042 as compared with 0.073) although the Fraser-Yancey efficiencies, yields, and quality of products were not too greatly at variance, is: The separation in test 26 was much sharper in the vicinity of the density of separation (1.65), indicating that there was less misplaced material of the 1.60—1.70 fraction in either product. However, the ash content of the coal in this density region is not very much different, so that if there were more 1.60-1.65 material in the refuse and more 1.65-1.70 material in the washed coal for one test than for the other, the final ash contents of the products would differ but slightly.

Examination of the data in Table I and the accompanying distribution curves in Figure 4 discloses that, once the specific gravity of suspension is chosen, the underflow apex diameter can be adjusted to give a separation at the required density. This separation will be quite sharp unless the yield of washed coal product is allowed to go above or below normal limits. For example, with the 1.30 medium, the cyclone can be adjusted to wash the coal at densities between 1.61 and 1.70 with yields ranging from 79.7 to 82.0 per cent with reasonable sharpness of separation, but when the cyclone underflow apex diameter is enlarged to the maximum opening of 0.82 inches, the yield drops to 55.0 per cent and the separation is made at a density of 1.43 with a resultant increase in probable error.

With the 1.40 suspension, the range of separation densities lies between 1.56 and 1.81 with yields ranging from 78.5 to 88.5 per cent. Using this medium, the cyclone gave reasonably sharp separations at all underflow apex diameters from the smallest to the largest opening. The runs with the 1.45 specific gravity medium showed that the sharpness of separation decreased as the underflow diameter was made smaller. This was evidence that, with this heavier suspension, the underflow apex opening must not be made too small if the probable error is to be kept at a low figure.

INFLUENCE OF SPECIFIC GRAVITY OF MEDIUM

The two variables of medium specific gravity and underflow apex diameter are so closely interrelated that it is difficult to discuss one without mentioning the other. This means that the setting of one of these variables fixes the other. The data of Table I have been rearranged in Table II in three series of tests at constant underflow apex diameters

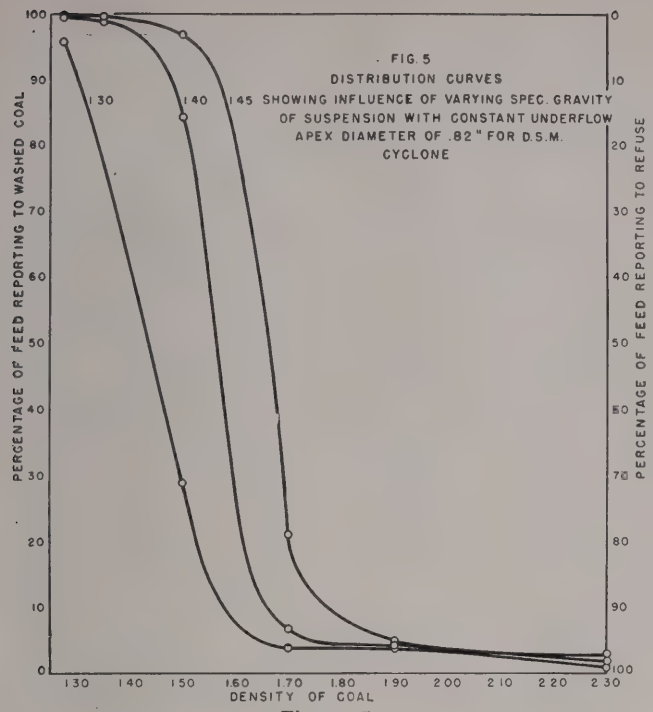

Figure 5.

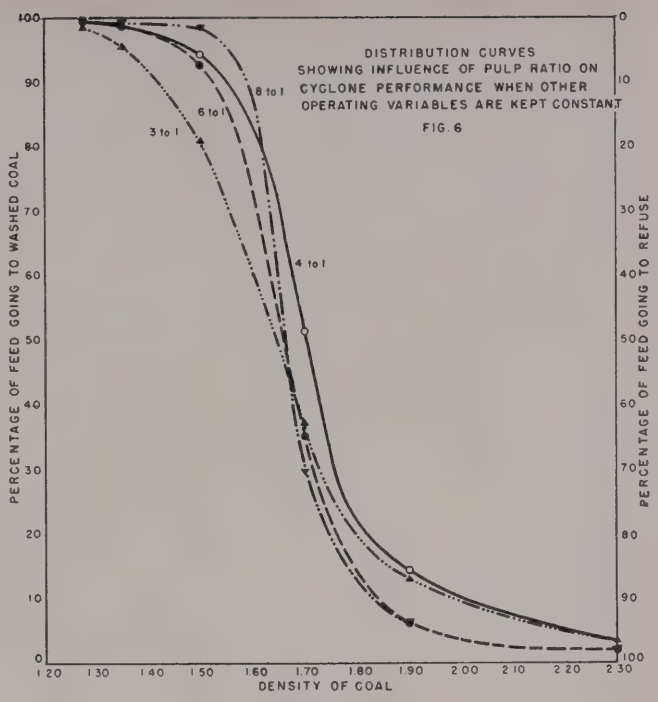

Figure 6.

with varying medium suspension specific gravities from 1.30 to 1.45. The distribution curves have been drawn for each series of tests at a constant underflow diameter. An example of the curves for one such series at a constant underflow diameter of 0.82 inches is shown in Figure 5. The curves show that, for fixed cyclone dimensions, the density of separation increases with heavier suspensions. This, of course, is what might be expected, since the density of the washing bed is increased with the heavier suspensions. If we again disregard the two high and low separation density values, an assessment of the data reveals that specific gravity of the medium suspension has little or no effect upon cyclone performance.

Influence of Pulp Ratio

The data in Table III and the distribution curves for these tests in Figure 6 show the influence of pulp ratio on cyclone performance. The runs were conducted under the following constant conditions: specific gravity of suspension 1.30, inlet feed pressure 13.5 p.s.i.g., inlet feed area 0.78 square inches, overflow vortex finder diameter 1.03 inches, and underflow apex diameter 0.59 inches. The densities of separation did not change much, ranging from 1.64 for the 3 to 1 pulp ratio to 1.70 for the 4 to 1 ratio. The best results were given by the 8 to 1 pulp ratio with a probable error of 0.054, while there was not much to choose between the performance at 6 to 1 and 4 to 1 ratios with probable errors of 0.070 and 0.072 respectively. The pulp ratio of 3 to 1 gave the large probable error of 0.115. These data give evidence that the smallest pulp ratio that should be used with the 1.30 suspension is in the neighbourhood of 4 to 1.

The fact that the sharpness of separation and output efficiency are not too greatly affected by the pulp ratio as long as this ratio is kept above a certain figure (3 to 1 in this instance) suggests that fluctuations in the feed rate to the cyclone would not greatly affect cyclone performance. This would mean that the heavy medium specific gravity would not have to be adjusted for a different rate of feed.

Effect of Density Distribution of Feed on Cyclone Performance

In an attempt to discover whether the density composition of the raw coal had any effect on cyclone performance, tests were conducted with feeds having various density distributions. It was desired to utilize feeds which differed only in the percentage of coal present in the several density fractions but which had, as nearly as

Table III.— Influence of Pulp Ratio

	Test No.			
	46	47	44	45
Pulp ratio	8 to 1	6 to 1	4 to 1	3 to 1
Yield of washed coal, %	82.0	82.2	84.2	79.4
Ash of washed coal, %	9.6	9.8	10.4	10.0
Ash of refuse, %	52.6	52.8	53.9	45.2
Calculated ash of feed	17.2	17.5	17.3	17.2
Density of separation	1.67	1.67	1.70	1.64
Probable error	0.054	0.070	0.072	0.115
Fraser-Yancey* efficiency, %	98.3	97.6	97.2	93.4

*See footnote to Table I.

The following conditions were constant for all the above tests with the three-inch-diameter cyclone:

Feed coal	Cadomin −10+100 mesh
Specific gravity of medium	1.30
Inlet feed pressure	13.5 p.s.i.g.
Feed inlet	0.78 in.×1.00 in.
Overflow vortex finder diameter	1.03 in.
Underflow apex diameter	0.59 in.

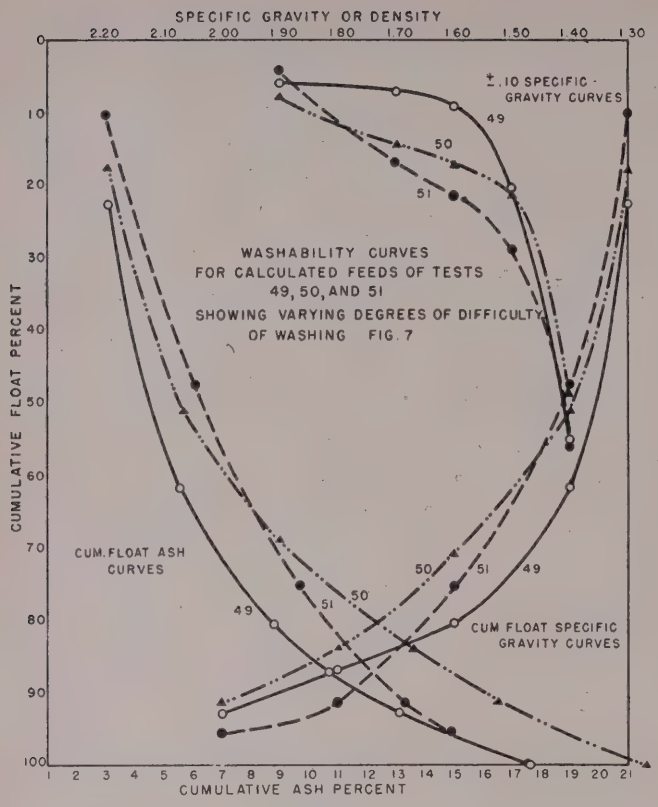

Figure 7.

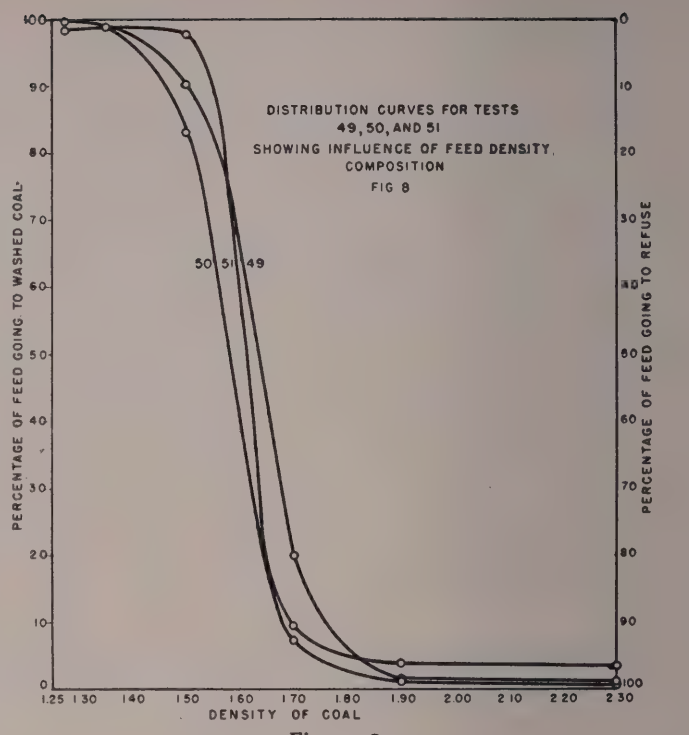

Figure 8.

possible, the same characteristics such as particle size, shape, etc. To achieve this end, the cyclone was used to wash Cadomin —10+100 mesh coal at different densities of separation. The products were blended with raw Cadomin —10+100 mesh coal to give the various feeds. So far as can be ascertained, no attempt of this sort has been made heretofore. Other workers have generally used different types of coal or added foreign material such as sand to the raw coal to obtain varying types of feed.

The washability curves for the feed coals of tests 49, 50, and 51 are shown in Figure 7. The feed for test 49 was the same as used for the previous tests, *i.e.*, Cadomin —10+100 mesh. The feed for test 51 had almost the same ash content* as that for test 49, 17.5 per cent as compared with 17.6 per cent. However, the ±0.10 specific gravity curve in Figure 7 reveals that the coal for test 51 was much more difficult to wash than that of test 49 at the densities of separation employed. The coal for test 51 has a ±0.10 specific gravity distribution of 21.0 per cent at 1.64 separation density, whereas the coal for test 49 has a ±0.10 specific gravity distribution of 8.0 per cent at its 1.63 separation density. The feed coal for test 50 contained more heavy material and had an ash content of 21.6 per cent. It can be seen from the ±0.10 specific gravity curve that the degree of washing difficulty of this coal was intermediate between that of the feeds for tests 49 and 51.

Bird has stated that if the ±0.10 curves gives a value of less than 7 per cent, the washing is comparatively simple and can be accomplished by almost any process. When the curve gives a value between 7 and 10 per cent, the washing is moderately difficult and requires an efficient process. The feed of test 49 lies within this category. Values of 10 to 15 per cent represent difficult washing, while percentages between 15 and 20 present a very difficult problem requiring an efficient process, low tonnages, and expert operation. The coal of test 50 falls within this category. The feed for test 51 with a

*All ash percentages quoted for the feed are calculated from the products.

TABLE IV.— EFFECT OF DENSITY DISTRIBUTION OF THE FEED

	TEST NO.		
	49	50	51
Yield of washed coal, %	79.7	68.1	75.8
Ash of washed coal, %	9.1	8.9	10.4
Ash of refuse, %	51.1	48.5	39.6
Calculated ash of feed, %	17.6	21.6	17.5
Density of separation	1.63	1.58	1.61
±0.10 sp. gr. distribution	8.0	17.6	21.0
Probable error	0.052	0.050	0.034
Fraser-Yancey* efficiency, %	97.7	95.2	96.0

*See footnote to Table I.

The following conditions were constant for all the above tests with the three-inch-diameter cyclone:

Feed coal	Cadomin −10+100 mesh
Specific gravity of medium	1.30
Pulp ratio	6 to 1
Inlet feed pressure	13.5 p.s.i.g.
Feed inlet	0.78 in. ×1.00 in.
Overflow vortex finder diameter	1.03 in.
Underflow apex diameter	0.66 in.

±0.10 specific gravity figure of 21.0 per cent is classed as an exceedingly difficult coal to wash.

The distribution curves for the series of runs are shown in Figure 8 and the resultant data for the three tests are given in Table IV. These runs were all made under constant conditions: specific gravity of suspension of 1.30, pulp ratio of 6 to 1, inlet feed pressure of 13.5 p.s.i.g., inlet feed area of 0.78 square inches, overflow vortex finder diameter of 1.03 inches, and underflow apex diameter of 0.66 inches.

The unblended feed of test 49 (with a ±0.10 specific gravity of 8.0 per cent) gave a probable error of 0.052, whereas the feed for test 50, which was a more difficult coal to clean (with a ±0.10 specific gravity of 17.6 per cent), gave almost the same probable error of 0.050. A surprising performance is illustrated by the data for test 51, in which the feed that was shown to be the most difficult to clean (with a ±0.10 specific gravity distribution of 21.0 per cent) was separated with the low probable error of 0.034. No explanation is advanced for the marked increase in sharpness of separation with this sample.

The above results demonstrate that density distribution of the feed has no detrimental effect on the sharpness of separation obtained with the cyclone. The Fraser-Yancey efficiencies are satisfactory for all three cleaning runs, indicating that there was little loss of saleable coal, although these efficiency values should not be compared since the densities of separation were not the same. These results illustrate the remarkable ability of the cyclone to cope with radical changes in the quality of the feed. This, coupled with its effectiveness in contending with varying pulp ratios, shows the cyclone to be surprisingly adaptable to changes in feed quality and feed rate. These factors would be quite significant in commercial cleaning plant operation.

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Contributed Discussion

Mr. G. A. Vissac (1): I wish to congratulate Mr. Morimoto on his very interesting paper. It represents a highly scientific and extensive effort to analyze one of the most promising new processes for the cleaning of fine coal.

I agree with Mr. Morimoto that, at the present time, there is really no satisfactory economical process available for the cleaning of bituminous coals below 1/8-inch.

I also feel that the use of centrifuge force, which can reach values several thousand times the value of gravity, offers attractive possibilities.

Unfortunately, the Dutch Cone, in order to differentiate the various gravities corresponding to the several products to be separated, must use a heavy medium suspension, and the subsequent necessity of separating and recovering this heavy suspension introduces difficult problems.

No practical and economical solution has yet been found for this separation of the heavy medium, and, as a consequence, the Dutch Cone is used today only as a thickener, separating the liquid on one side from all the solids on the other side.

We believe a more practical solution can be reached without the use of a heavy medium suspension, by differential amplitudes of the vertical components of the centrifugal forces involved; we are working on such a solution.

In all future investigations, however, Mr. Morimoto's work will afford an excellent and valuable guide for the development and control of processes for the cleaning of fine coal, using centrifugal forces.

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The Aqueous Oxidation of Pyrite*

By JAMES F. STENHOUSE† and W. M. ARMSTRONG§

(*Annual General Meeting, Quebec City, Que., April,* 1951)

(*Transactions, Volume LV,* 1952, pp. 38-42)

ABSTRACT

The aqueous oxidation of pyrite in caustic solutions under oxygen pressure was investigated by measuring the effect of reaction variable on oxygen consumption by the reactions. From the results of this investigation, a model for the process has been proposed.

In this model, sulphur diffuses as a positive ion through an oxide layer which forms on the sulphide particles and is then oxidized to sulphate ion. Iron is oxidized by substitutional diffusion of oxygen atoms into the oxide layer. The effect of most reaction variables on the rate of oxygen consumption can be explained with this model.

Introduction

RECENT investigations (1,2,3) have shown that certain sulphide ores can be readily leached by agitating the ore in the form of a pulp under pressure in the presence of oxygen. Since very little was known of the mechanism of this oxidation, a programme of investigation was inaugurated in 1949 in the laboratories of the Department of Mining and Metallurgy of the University of British Columbia. Financial support for the investigation was received from the National Research Council in the form of student grants and bursaries, from the Canadian Liquid Air Company, Limited, and from the Consolidated Mining and Smelting Company of Canada, Limited. The first objective was to devise a technique to study the primary oxidation reactions in aqueous solutions and to apply the technique to a detailed study of the oxidation of pyrite in caustic solutions.

Pyrite was chosen for the experimental work not only because it is a common constituent in many base metal ores but also because, when pyrite is oxidized in caustic solutions, the reaction resembles that of the roasting of sulphides in a gas atmosphere.

Experimental Work

Description of Reactions

The reactions were carried out in a rocking-type autoclave. The pyrite, which had been screened into particle size ranges and measured (4), and the caustic solution, were placed in the reaction chamber. The chamber was then sealed, heated, and agitated. When the reaction temperature was reached, oxygen was fed from an oxygen storage bottle into the reaction chamber. A constant total pressure was maintained in the chamber with an oxygen regulator in the oxygen line feeding the reaction chamber.

The final products of the aqueous oxidation of pyrite in caustic solutions are iron oxides and sulphate ion. The iron oxides form a stable layer around the sulphide particles. Both Fe_2O_3 and Fe_3O_4 lines were found in x-ray patterns of oxide residues from completely and incompletely oxidized charges. The intensity of lines in the patterns indicated that a relatively large amount of amorphous material was present.

Sulphur is oxidized to sulphate ion. This ion must be formed through a series of reactions between sulphur and oxygen although, in all stages of the reaction, the concentration of intermediate products in the series was too low to be determined by chemical analytical methods.

Method of Investigation

The investigation was based on the determination of the effect of the following reaction variables on the rate of oxygen consumption by the reactions:

(1) Average particle size of pyrite charge
(2) Partial pressure of oxygen above the reaction
(3) Temperature of oxidation
(4) Caustic concentration of solutions

The effect of silica in the charge and the effect of potential on the reaction rates were investigated qualitatively.

Oxygen consumption was measured by recording oxygen pressure in the storage bottle feeding the reactions and correcting this record for error caused by fluctuation in the oxygen partial pressure in the reaction chamber. These corrections were calculated from a record of reaction chamber pressure and temperature.

Pressure was determined by recording the potential developed across a wheatstone bridge of nichrome wire built into a bourdon tube gauge connected to the oxygen supply bottle. Changes in the pressure of oxygen in the supply bottle caused changes in the tension in two legs of the bridge and developed a potential across the bridge. The change in potential was proportional to change in pressure.

Analysis of Oxygen Consumption Curves

From the oxygen consumption curves, the logarithm of the difference between total oxygen consumption and consumption at definite time intervals was calculated and plotted against time (Figure 1). The latter portion of this logarithmic plot was exponential. For runs made in caustic solutions below two molar, this exponential portion of the curve was extrapolated to the zero time axis (curve *B*). The logarithm of the difference between the oxygen consumption necessary to produce such a curve and the original curve was calculated and

* Part of this research was supported by a National Research Council Bursary.

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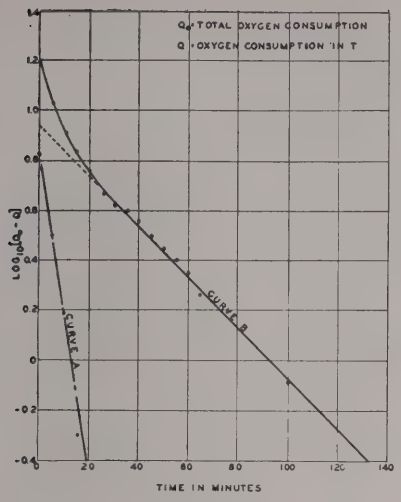
1, 2, 3,) For references, see end of paper.



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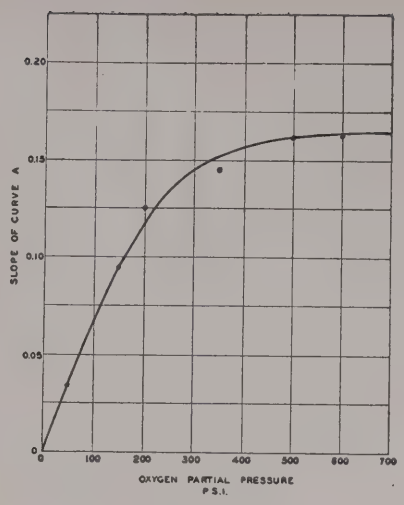

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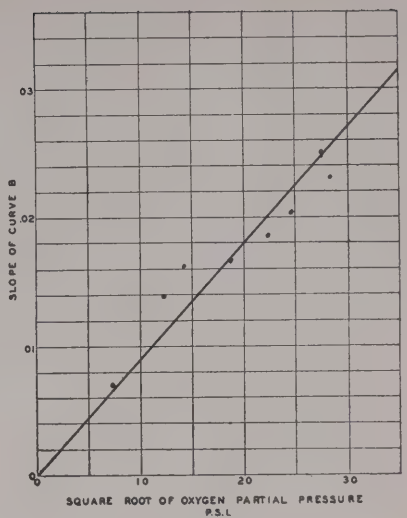


Figure 3.

plotted against time (curve *A*). Curve *A* was exponential if oxygen partial pressure was above a critical value. Below this value, the slope of the curve increased with time.

Runs made in caustic solutions above two molar could not be divided into two exponential curves. The slope of the initial portion of these oxygen consumption curves was constant. The latter portion was exponential.

Effect of Reaction Variables

All variables investigated by measuring oxygen consumption were varied independently from the following conditions:

CHARGE TO UNIT

40 grams pyrite
1,500 milliliters water
60 grams NaQH

CONDITIONS

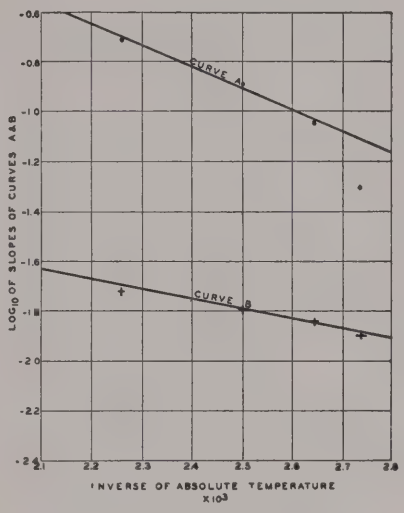
Temperature, 120°C.
Oxygen partial pressure, 200 p.s.i.
Average particle size of pyrite charge, 10.9 microns

The effect of changes in pressure, temperature, and average particle size of pyrite charge on the slopes of curves *A* and *B* (Figure 1) was determined and plotted in Figures 2, 3, 4, and 5. If curve *A* was not exponential, the slope of the initial portion of the curve was plotted.

The effect of caustic concentration on the oxygen consumption curves is given in Figure 6.

The silica content of the charge had an effect on the rate of oxygen consumption by the reactions. With the addition of 40 grams of silica flour to the above charge, oxygen consumption was not measurable after 30 minutes of reaction time.

Potential also had an effect on the rate of oxidation. A single pyrite crystal, approximately a ½-inch cube, was placed within a 1½-inch-diameter cylindrical shield in the reaction chamber and a 6-volt potential was applied between the crystal and the shield, with the crystal as the cathode. A second crystal of approximately the same size was also placed in the reaction chamber. Oxidation was carried out at 130°C. and 100 p.s.i. oxygen pressure in a one molar caustic solution for four hours. The cathodic crystal was completely oxidized in this time while the second crystal showed only slight surface tarnish.



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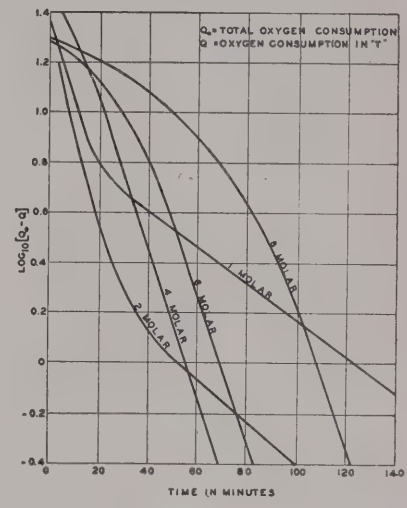

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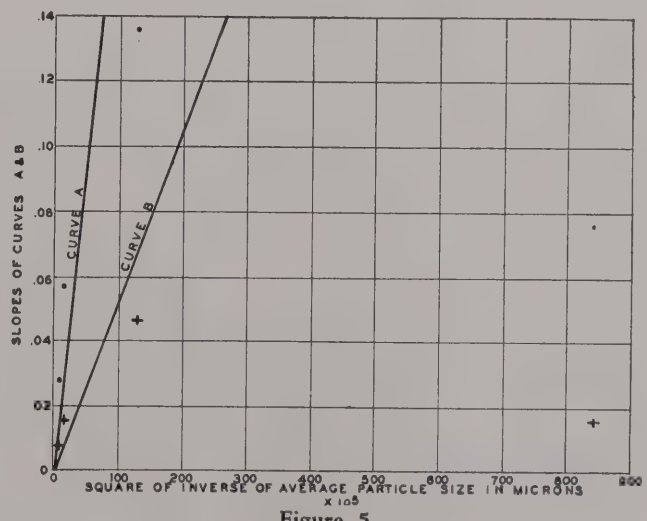

Figure 5.

Interpretation of Experimental Results

Model for the Oxidation Process

During the aqueous oxidation of pyrite in caustic solutions, a stable oxide layer forms on the surface of the pyrite particles. Oxidation must take place across this growing oxide layer. The following steps are proposed for this oxidation process:

(1) Adsorption of oxygen molecules to the surface of the particle and dissociation of the molecules to atoms on the surface
(2) Transfer of electrons from sulphur atoms, through the oxide layer, to adsorbed oxygen atoms
(3) Diffusion of sulphur as positive ions through the oxide layer to the surface of the particle
(4) Reaction between sulphur ions and oxygen on the surface of the particle
(5) A series of reactions involving this sulphur-oxygen compound and oxygen to form sulphate ion
(6) Diffusion of lattice defects, created at the pyrite-oxide interface by sulphur removal, to the surface of the particle and diffusion of oxygen atoms into the oxide layer

The problem of deriving a mathematical expression for the rate of oxidation based on this model involves the problem of diffusion from a moving interface. Since no general solution for this problem is known (5), it was necessary to base the derivation on a simplified model.

Expression for the Rate of Diffusion of Sulphur

In the derivation of the expression for the rate of diffusion of sulphur, the interface reaction has been neglected. Sulphur is assumed to diffuse from a hypothetical sphere. At zero time, the concentration of sulphur ions throughout this sphere is constant. The concentration of sulphur ions at the surface of the sphere is zero for all time. Under these assumptions, the solution is equal to

$$Q = Q_0 - \frac{8 C_0 r^3}{\pi} e^{-\frac{\pi^2 D}{r^2} t}$$

where

Q = Quantity of sulphur diffused from the sphere in time t
Q_0 = Initial quantity of sulphur in the sphere
C_0 = Initial concentration of sulphur in the sphere
r = Radius of the sphere
D = Diffusion coefficient
t = Time.

This expression is based on the assumption that the concentration of sulphur at the surface of the particle is zero. This assumption is inexact since, in the reaction between sulphur and oxygen on the surface of the oxide layer, there will be an equilibrium concentration of sulphur which will depend upon the rate at which sulphur arrives at the surface and the rate at which it reacts with oxygen.

From equation (1), the expression for the rate at which sulphur arrives at the surface of the particle at constant temperature has the form

$$Ae^{-kt}$$

where

A and k = constants
t = time

The expression for the rate of reaction of sulphur at the surface of the particle will depend upon the concentration of oxygen adsorbed to the surface and the concentration of sulphur on the surface. The expression will also depend upon the product of this surface reaction. For simplicity, assume that one atom of oxygen reacts with one atom of sulphur. The expression for the rate of this surface reaction at constant temperature will then have the form

$$KC_0C_s$$

where

K = constant
C_0 = concentration of oxygen
C_s = concentration of sulphur

The expression for the change of the concentration of sulphur on the surface at constant temperature is equal to

$$\frac{dC_s}{dt} = Ae^{-kt} - KC_0C_s$$

At constant pressure, the concentration of adsorbed oxygen is assumed constant. Solution of this linear first-order differential equation yields

$$C_s = \frac{A}{KC_0 - k}\left(e^{-KC_0t} - e^{-kt}\right)$$

When the concentration of adsorbed oxygen is large, KC_0 approaches k and the expression reduces to zero, and the assumption made in the derivation of the expression for oxidation of sulphur, *i.e.*, that the concentration of sulphur at the surface of the particle is zero, will not be seriously in error. At low oxygen pressures, however, the assumption will be more seriously in error. The concentration of adsorbed oxygen will be low and the expression for the concentration of sulphur on the surface will be given by equation (2). A sketch of this function is given in Figure 7.

The total change in concentration of sulphur on the surface of the particle with time will increase with decreasing pressure. This relatively high concentration of sulphur will have an important effect on the rate of oxidation of the sulphur. Effectively, it will decrease the concentration gradient of sulphur through the oxide layer and thus reduce the rate of diffusion of sulphur. At low oxygen pressures, the concentration of sulphur at the surface of the particle will follow the curve shown in Figure 7. The concentration of gradient for sulphur, compared to the gradient in a particle with zero concentration at the surface, will decrease to a minimum at C_m and then increase with time. The rate of diffusion of sulphur will also decrease to a minimum at C_m and then increase with time.

In the region of C_m (Figure 7), the concentration of sulphur adsorbed to the surface of the particle will be substantially constant. In this region, the rate of oxidation of sulphur should be very close to an exponential.

The product of the surface reaction between sulphur and oxygen is oxidized to sulphate ion through a series of reactions. Experimentally, the concentrations of intermediate products of this series were too low to permit identification of the compounds by chemical analytical methods. Under the condition of low concentrations of intermediate compounds, the diffusion of sulphur to the surface of the particle must be the rate controlling reaction in the series and the rate of oxygen consumption by all the reactions will be proportional to the rate of diffusion of sulphur.

Expression for the Rate of Diffusion of Oxygen into the Oxide Layer.

The expression for the rate of diffusion of oxygen into the oxide layer is derived on assumptions similar to those for diffusion of sulphur. Oxygen is assumed to diffuse into a hypothetical sphere with an initial oxygen concentration equal to zero. The concentration of oxygen on the surface of the sphere is assumed to be constant. Under

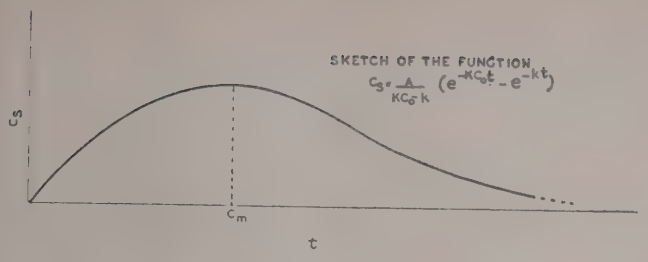

Figure 7.

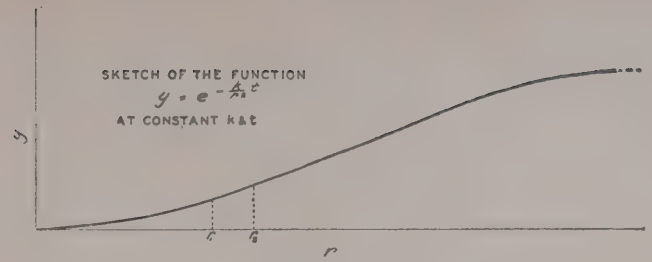

Figure 8.

these assumptions, the expression is equal to

$$Q = Q_0 - \frac{8 C_0 r^3}{\pi} e^{-\frac{\pi^2 D}{r^2} t}$$

where

Q = Quantity of oxygen diffused into the sphere in time t

Q_1 = Final quantity of oxygen in the sphere

C_1 = Concentration of oxygen at the surface of the sphere

This expression neglects possible effects of change of phase in the oxide and the effect of concentration of lattice defects in the sphere.

Effect of Particle Size Distribution

The mathematical expressions for the oxidation of iron and sulphur are derived on the basis of oxidation of a single particle. The experimental oxygen consumption curves were obtained from the oxidation of pyrite charges with unknown particle size distributions. In order to compare the theoretical and experimental results it is necessary to estimate the effect of a particle size distribution on the mathematical expressions.

The mathematical expressions for the oxidation of iron and sulphur have the form

$$A r^3 - B r^3 e^{-\frac{K}{r^2} t}$$

where

A, B, and K = constants

r = radius of particle

t = time

The expression for the oxidation of n such particles with a particle size distribution given by $n = g(r)$ will be equal to

$$A\int_0^\infty r^3 g(r)\,dr - B\int_0^\infty r^3 g(r)\, e^{-\frac{K}{r^2} t}\, dr.$$

No solution for this type of problem could be found in a review of mathematical literature on statistics and it was therefore necessary to apply approximation methods to the problem.

Consider a plot of the term

$$e^{-\frac{K}{r^2} t}$$

This expression has the general form shown in Figure 8. If the particle size distribution has definite limits, r_1 and r_2, and r_a is the average particle size of this distribution, the expression (4) may be assumed to be a constant equal to

$$e^{-\frac{K}{r_a^2} t}$$

The error in this assumption will be small providing the slope of the curve (Figure 8) is small in the range $r_1 \leq r \leq r_2$.

The expression for the total oxygen consumption by the charge will then reduce to

$$A\int_0^\infty r^3 g(r)\,dr - B e^{-\frac{K}{r_a^2} t}\int_0^\infty r^3 g(r)\,dr$$

The expression

$$\int_0^\infty r^3 g(r)\,dr$$

is equal to the third moment of the distribution about the origin multiplied by the total number of particles in the charge:

$$Q = NA\bar{r}^3 - NB\bar{r}^3 e^{-\frac{K}{r_a^2} t}$$

The error in this expression decreases with decreasing spread of the distribution, decreasing average particle size, and increasing time.

No abnormalities in the experimental oxygen consumption curves that could be attributed to particle size distribution were encountered in the investigation and therefore it is assumed that errors from this source were too small to be measured.

Identification of Experimental Curves

The mathematical expression for the oxygen consumption by a pyrite charge under a high oxygen partial pressure is composed of the sum of two exponential expressions, one for the oxidation of sulphur, the second for the oxidation of iron. Analysis of experimental oxygen consumption curves yielded the same result (Figure 1). The curve A (Figure 1) is the curve for the oxidation of sulphur. This fact was established experimentally by analysis of residues taken from incompletely oxidized charges. Complete sulphur removal from the residues coincided with low logarithmic values for curve A. Curve B must therefore correspond to the oxidation of iron.

The oxidation of iron and sulphur are not completely independent. The oxidation of iron depends upon diffusion of oxygen into the oxide layer. This diffusion is related to the concentration of lattice defects in the layer and thus to sulphur removal.

Effect of Reaction Variables on Model Oxidation and Comparison with Experimental Results

Potential

Positive sulphur ions are formed at the pyrite-oxide interface by transfer of electrons from the sulphur to adsorbed oxygen atoms on the surface of the particle. Assuming that this process of electron transfer is in equilibrium, the number of sulphur ions formed will depend upon the difference in potential between electron levels in the pyrite and vacant sites on adsorbed oxygen atoms (6).

The application of a potential across the oxide layer on a pyrite crystal with the pyrite crystal as the cathode will increase this difference in potential and thus increase the number of sulphur ions formed. The increased concentration gradient of sulphur ions across the oxide layer will increase the rate of diffusion of sulphur. The application of a potential will also increase the rate of oxidation of iron through the formation of lattice defects.

Experimentally, application of a potential increased the rate of oxidation of a single pyrite crystal.

Pressure

Curve A (Figure 1), which is the curve for oxidation of sulphur, approaches exponential as oxygen partial pressure increases. Experimentally, the curves become exponential when oxygen partial pressure exceeds 400 p.s.i. at 120°C. This

condition is in agreement with the model for oxidation where increasing the oxygen partial pressure causes a reduction in the concentration of sulphur on the surface of the particle, and the expression for oxidation approaches the exponential relation given in equation (6).

The experimental effect of pressure on the rate of oxidation of sulphur is given in Figure 2. This is a saturation type curve which is in agreement with the model for oxidation. The rate of oxidation of sulphur becomes independent of pressure as C_m (Figure 7) approaches zero.

The experimental rate of oxidation of iron is proportional to the square root of the oxygen partial pressure (Figure 3). This relation indicates that the oxidation process involves surface dissociation followed by diffusion, and these steps have been proposed in the model for the oxidation process.

Temperature

The experimental relations between temperature and the rates of oxidation of iron and sulphur are given in Figure 4. These results were obtained from runs made with oxygen partial pressure below that required to yielded an exponential relation for the oxidation of sulphur. The effect of temperature on the rate of oxidation of sulphur will therefore depend not only upon the activation energy for the diffusion process but also upon the sensitivity of the constants in the expression for the surface concentration of sulphur to temperature. If the product KC_0 is more sensitive to temperature than is k in equation (2), low temperature rates would be below those expected from an Arrhenius-type equation. This condition was encountered in the oxidation of sulphur (curve *A*, Figure 4). The experimental activation energy obtained by joining higher temperature results is 4,000 calories.

A better experimental curve was obtained for the effect of temperature on the oxidation of iron (curve *B*, Figure 4). The experimental activation energy for this process is 2,000 calories.

The low experimental values for the activation energies is believed to be caused by the state of disorder that must exist in the oxide layer.

Particle Size

Based on the mathematical expressions for the oxidation of iron and sulphur, the rates are inversely proportional to the square of the average particle size. The experimental plot is given in Figure 5. The deviation from the theoretical results is believed to be caused by variations in the silica content of the charge. The analysis of the charges was:

Av. Particle Size, Microns	% Fe	% S	% SiO_2
10.9	44.6	47.0	5.0
27.7	47.5	50.4	1.8
88.3	48.2	50.8	0.8
119.0	48.2	51.0	1.0

The amount of deviation from the theoretical law increased with silica content.

Caustic Concentration

Increasing the caustic concentration from one to two molar increased the experimental rate of oxidation of sulphur and iron. Above this concentration, the reactions became controlled by some other step and it was not possible to obtain curves for the oxidation of sulphur. In these cases, the initial rate of oxygen consumption by the reactions became constant.

Increasing the caustic concentration also increased the rate of oxidation of iron.

These increases in the rates of oxidation with caustic concentration cannot be explained by the model, but it is believed that they were caused by reactions between caustic and silica.

Conclusions

(1) Pyrite oxidizes at a measurable rate at lower temperatures in caustic aqueous solutions than in air.

(2) Experimental oxygen consumption curves are composed of the sum of two curves. One curve is attributed to the oxidation of sulphur, the second to the oxidation of iron. The rates of oxidation of these materials exhibit a high degree of independence.

(3) The curves, and the effect of most reaction variables on these curves, can be explained by assuming that sulphur diffuses through the growing oxide layer on the sulphide particles as a positive ion. The oxidation of iron is assumed to take place by substitutional diffusion of oxygen into the layer.

(4) The utilization of the rate of oxygen consumption affords a satisfactory method for studying aqueous oxidation reactions. Experimental results are consistent with well known physical laws. This fact indicates that the method is basically sound. The main disadvantage of the method lies in the difficulty in maintaining the unit free of leaks at high pressures.

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A Nomenclature of Igneous Rocks*

By I. C. BROWN†

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Introduction

MANY TABLES of nomenclature for igneous rocks have been introduced from time to time, and each has its strong proponents. As a result, any discussion of igneous rocks is apt to prove confusing unless the names used for these rocks are defined. This became so obvious to a discussion group, formed by the members of the Geological Survey of Canada in 1948, that an attempt was made to establish a scheme of nomenclature that could be used by any member mapping anywhere in Canada. The writer was asked to co-ordinate the ideas presented by individuals in the discussion group, to arrange them in tabular form, and, later, to distribute copies of the resultant table to Canadian universities, provincial geological surveys, and some other geological discussion groups. Many useful suggestions and criticisms were received, were studied by members of the original group, and, so far as practicable, were incorporated in the scheme of nomenclature. The result is the table presented here.

This nomenclature of igneous rocks is at present being used by most members of the Geological Survey of Canada who are working with igneous rocks, and by some Canadian universities. We hope it will be found useful and be more widely accepted. With this idea in mind, certain principles were adopted in drawing up the nomenclature. These are:

(1) The rocks should be named solely on the basis of mineralogical constitution and physical appearance.

(2) The names should be, so far as possible, in general use by prospectors, miners, and geologists.

(3) The table should be simple, with as few divisions as practicable.

(4) The nomenclature should provide a basis for more specific rock names, and these should not replace well established local rock names.

(5) The names should be applicable, so far as possible, both in the field and after petrographic study.

The attempt to adhere to these principles has resulted in a table of nomenclature that is at once a compromise and simplification of existing classifications, and is believed to be best suited for general usage.

Explanation of Table

General

This nomenclature is intended to be used in naming any igneous rock, or any rock which in the worker's opinion has an igneous texture.

The explanations given below deal with the divisions of the table and the reasons these divisions were chosen. They are intended to be read in conjunction with the table. More general explanations will be found in the next section of the paper.

Amount of Quartz

In most classifications, the amount of quartz present is stated in general terms only, whereas in others it is given in percentages that vary from 5 to 33. It was decided to use 10 per cent as the dividing line between rocks essentially "with quartz" and "without quartz" because that amount is normally easily seen in any rock specimen whereas a smaller amount is only noticeable with some difficulty. Also, many workers estimate percentages and it was found that 10 per cent could be more accurately estimated than the commonly used 5 per cent.

Proportion of Potash Feldspar to Total Feldspar

The division is threefold to agree with most classifications. Some workers, such as Holmes, divide into sixths but this was considered too fine, and led to too complicated a table.

Percentage of Feldspar

The 10 per cent dividing line was decided on for the same reasons as those offered in the case of quartz.

Composition of Plagioclase

The dividing lines in the composition of the plagioclase feldspar are taken directly from the commonly used tables for determination of plagioclase feldspars. It was suggested that the division be made where the X index of refraction of plagioclase equals the index of refraction of balsam, which is at An_{17}, but it was decided that the better division is at An_{10}, where the Y or median index of refraction equals that of balsam.

Aphanitic or Fine-Grained Rocks

The general trends for fine-grained rocks are included in smaller type below their coarse equivalents. They are intended for use where the rock is too fine grained for megascopic determination, and the dividing line is left to the worker's judgment. In cases where the composition cannot be determined, the more general terms 'felsite' and 'trap' are used on a colour basis only. The term 'trap' was decided on rather than the term 'mafite' because the former is more widely used.

Content of Dark Minerals

The common content of dark (mafic) minerals is added to serve as a

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NOMENCLATURE OF IGNEOUS ROCKS

Amount of Quartz	Proportion of Potash Feldspar to total Feldspar	More than 10% Feldspar — Composition of Plagioclase: Albite An_0 — An_{10}	Oligoclase An_{10} — An_{30}	Andesine An_{30} — An_{50}	Labradorite, etc. An_{50} — An_{100}	Less than 10% Feldspar	Monomineralic — Commonly less than 10% extraneous minerals
More than 10% Quartz	More than 2/3	GRANITE *RHYOLITE*					
	1/3 - 2/3		QUARTZ MONZONITE *QUARTZ LATITE*				
	Less than 1/3	ALBITE GRANITE *RHYOLITE*	GRANODIORITE *QUARTZ LATITE*	QUARTZ DIORITE *DACITE*	QUARTZ GABBRO *QUARTZ BASALT*		
Less than 10% Quartz	More than 2/3	SYENITE *TRACHYTE*				PERKNITE PERIDOTITE (More than 5% Olivine)	PYROXENITE HORNBLENDITE DUNITE ANORTHOSITE
	1/3 - 2/3		MONZONITE *LATITE*				
	Less than 1/3	ALBITE SYENITE *TRACHYTE*	SYENODIORITE *LATITE*	DIORITE *ANDESITE*	GABBRO *BASALT*		

COMMON CONTENT OF DARK MINERALS

0 — 10	10 — 40	40 — 70	70 — 100

GENERAL TERMS FOR FINE-GRAINED ROCKS

FELSITE *(light coloured on fresh surface)* TRAP *(dark coloured on fresh surface)*

Geological Survey of Canada

guide in naming rocks in the field where, due to the difficulty of identifying the feldspars, this is done partly on a colour basis.

General Explanation

Local Terms

This nomenclature is not intended to supersede local or more specific rock names. It is intended that, in the description of rocks in a report, they should first be identified from the table, using the names supplied, which are well known everywhere. Subsequently, more specific or local names may be applied and used throughout the rest of the report. For example, in referring to the Sudbury norite, one could say, "the rock is a hypersthene *gabbro*, commonly called norite, and it will be referred to as norite in this report"; or, referring to some of the rocks in the Monteregian hills, "Rougemont is composed largely of *gabbro* of the following composition . . . This rock was originally named rougemontite, and will be referred to by that name to avoid confusion with other similar gabbros".

Textural and Genetic Terms

Textural terms are used as prefixes with the rock name. For example, a gabbro that shows diabasic texture is called a 'diabasic *gabbro*'.

Porphyritic rocks are treated by prefixing the term 'porphyritic' to the rock name, as 'porphyritic granite'. If the composition of the phenocrysts is important it is entered in parentheses after the term porphyritic as, 'porphyritic (quartz-orthoclase) rhyolite'. In this case both quartz and orthoclase phenocrysts are present and there is more orthoclase than quartz. Porphyries are named on the bulk mineral composition rather than on the composition of the phenocrysts. For example, the common 'quartz porphyry' could have a light rhyolitic or a dark dacitic groundmass and this is indicated by the name 'porphyritic (quartz) *rhyolite*' or 'porphyritic (quartz) *dacite*'.

The nomenclature is entirely non-genetic in so far as the actual rock name is concerned. Thus a rock of unknown origin from an isolated outcrop can be named. However, when the origin is known or inferred, it should be explained along with the name. For example an outcrop of andesite might be an *andesite* flow, an *andesite* sill, or an *andesite* dyke, or, a granitic rock might be an intrusive *granite* or a *granite* formed by processes of granitization.

Further Subdivision

More precise rock names than those shown in the table are obtained by using varietal minerals as prefixes. The varietal mineral prefixes, if more than one, are hyphenated and arranged in increasing order of importance. Rock names are not hyphenated. For example, a granodiorite with a notable amount of biotite and minor hornblende is a 'hornblende-biotite *granodiorite*', and a quartz monzonite with a similar content of accessory minerals is a 'hornblende-biotite *quartz monzonite*'.

Flexibility

The nomenclature is flexible in that rocks falling near or on dividing lines may be moved to either side, depending on which name the worker considers most representative. For example, suppose that samples from an intrusive mass carried from one-sixth to slightly more than one-third of the total feldspar as orthoclase, and that the plagioclase feldspar ranges from An_{25} to An_{40} with the average An_{35}. The names *quartz monzonite, granodiorite,* and *quartz diorite* would have to be used if the divisions of the table were strictly adhered to, but the worker might decide that, as all

samples were dark coloured, the name *quartz diorite* was most representative, and would, therefore, apply that name to the whole intrusive body.

Also, the nomenclature is flexible in that any igneous-textured rock, such as some metamorphic rocks, may be named directly or by recasting to original minerals. When naming a metamorphic rock the prefix 'meta-' should be used, or some explanation given to indicate that the rock is not strictly igneous.

Colour

Conspicuously light-coloured rocks are indicated by the prefix 'leucocratic' or 'leuco-', and dark-coloured ones by 'melanocratic' or 'mela-'. The names of any other rocks with unusual colour characteristics are prefixed by that of the appropriate colour. Colour prefixes are based on the colour of the fresh surface.

Granite and Syenite

These terms fall into a large number of pigeon holes in the table, but some of the rocks are extremely rare. All rocks in which more than two-thirds of the feldspar is potash feldspar are included under this heading, and if an unusual feldspar, such as bytownite, is present it is treated as a varietal mineral. Rocks in which two-thirds or more of the feldspar is albite are called *albite granite* or *albite syenite*. 'Albite' is used rather than 'soda' because the mineral prefix is considered to be more in agreement with the principles of the table.

For example, a rock with 60 per cent albite, 15 per cent orthoclase, 20 per cent quartz, and 5 per cent biotite is a 'biotite *albite granite*', whereas a rock with 60 per cent orthoclase, 15 per cent oligoclase, 20 per cent quartz, and 5 per cent biotite is a 'biotite-oligoclase *granite*'. In the first case, the varietal mineral is biotite and *albite granite* is the rock name, and in the second case biotite and oligoclase are the varietal minerals and *granite* is the rock name.

Ultramafic and Monomineralic Rocks

The ultramafic rocks are grouped under two names, 'perknite' if there is less than 5 per cent olivine, and 'peridotite' if there is more than 5 per cent olivine. More detailed names are obtained by using prefixes or existing terms.

Monomineralic rocks are named by the commonly accepted terms. 'Hornblendite' is preferred to the more general 'amphibolite' as the latter is in general use for metamorphic rocks.

Feldspathoids

A separate division based on feldspathoids is not used as these rocks are not widespread. They are named by using the prefix 'nepheline' and further subdivision is by use of local terms.

For example, a syenite with 20 per cent dark minerals and 25 per cent nepheline is a melanocratic nepheline syenite.

Grain Size

Grain size limits are as follows:

Coarse ..... greater than 5 mm.
Medium ..... between 5 mm. and 1 mm.
Fine ..... less than 1 mm.

Grain sizes given apply to the largest grains in the groundmass.

Glasses

True glasses are comparatively rare rocks, and are named in accordance with any of the existing terminologies.

Order of Prefixes

Prefixes should be kept to a minimum, but where more than one prefix is necessary the following order is suggested: grain size, texture, varietal minerals, rock name. Normally, only one or two outstanding characteristics will be included as prefixes and the rest left to the detailed description of the rock.

Rock Names Not In The Table

Some commonly used rock names, not included in the table, are intended to be used as described below.

Diabase. — Primarily a textural term and secondarily a rock name.

Lamprophyre. — Dark basic rocks, mostly occurring as dykes. They are composed predominantly of mafic minerals and commonly are porphyritic, with phenocrysts of the dark silicate minerals.

Pegmatite. — A very coarse rock usually irregular in grain and commonly characterized by graphic intergrowths of quartz and feldspar. Most pegmatites are mineralogically similar to granite, but pegmatitic equivalents of gabbro and diorite are known.

Aplite. — A leucocratic, sugary grained, dyke rock composed almost entirely of quartz and feldspar with commonly a little white mica.

It is possible that there are rocks whose composition would put them in the two right-hand columns of the table but which are not covered by the names given there. So far as known there are no commonly accepted terms for such rocks and consequently the space is left blank.

Acknowledgments

It would be impossible to list individually all those who have contributed to the development of this rock nomenclature. Thanks are due particularly to the members of the two discussion groups of the Geological Survey of Canada who gave freely of their time. The many useful suggestions and criticisms submitted by university staffs, provincial geological surveys, discussion groups, and numerous mining men and geologists, were of great help.

Comments on this rock nomenclature are invited. They should be sent to the author at the Geological Survey of Canada, Ottawa.

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Mining at Quemont

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(*Transactions, Volume LV,* 1952, pp. 47-58)

Introduction

THIS PAPER will describe the mining plant and underground operations at Quemont. Details of geology and milling are beyond its scope. Papers on these subjects will be prepared and presented separately. A detailed account of the exploration and development, 1944 to 1947, has been given by W. L. Brown (1).

Location

The mining concessions of Quemont Mining Corporation, Limited, are in Rouyn township, Rouyn-Noranda county, northwestern Quebec. These concessions total about 640 acres in area. They lie within the limits of the City of Noranda, contiguous to, and directly north of, the Horne mine of Noranda Mines, Limited. A large area in the southern part of the property is covered by the north arm of Osisko lake.

Production

The present operating rate is about 2,100 tons per day. Three flotation concentrates are produced: copper, containing a large proportion of the gold and silver in the ore; zinc; and pyrite, which carries an appreciable additional proportion of gold. The copper and zinc concentrates are shipped to smelters as produced. The pyrite concentrate is cyanided to recover additional gold, after which it is filtered, dried, and shipped for the manufacture of sulphuric acid. The principal revenue metals are copper, gold, and zinc. Silver and pyrite are important by-products.

*Manager, †Mine Superintendent, §Mine Production Engineer, Quemont Mining Corporation, Limited, Noranda, Que.

(1) *History of the Development of Quemont Mining Corporation, Limited;* The Northern Miner, March 6th, 1947.

History

The property was originally known as the Murray Group. Prior to 1926, surface exploration and some diamond drilling had been done. In that year the United Verde Extension Company, Arizona, optioned the group. This Company sank a two-compartment shaft, established a level at approximately 200 feet, and completed about 3,000 feet of exploratory driving from which 6,000 feet of diamond drilling was done. This option was dropped. In 1928, the Mining Corporation of Canada, Limited, purchased a large interest, and Quemont Mining Corporation, Limited, was formed to operate the property.

The shaft was deepened to 922 feet, and exploration levels were established at 500-foot and 900-foot horizons approximately. Lateral driving and diamond drilling on these levels totalled 4,800 feet and 3,000 feet respectively. Operations were stopped in September, 1930, because no ore had been indicated. For thirteen years nothing was done on the property. The development plant and equipment at the exploration shaft were dismantled and sold during the 1930's.

An exploration programme, consisting of a magnetometer survey and diamond drilling from surface, was begun in 1944. In February, 1945, No. 10 hole was drilled from the ice on Osisko lake. This hole intersected a massive sulphide body carrying appreciable amounts of gold, copper, and zinc. Two more intersections were obtained beneath the lake before the spring break-up, after which drilling was continued from the shore-line. It was not possible to direct these shore-line holes into the area under the lake at satisfactory angles. However, many ore intersections were obtained, which indicated the probability of large tonnages, but gave little indication of the outline or attitude of this ore.

Meanwhile, preparations for underground exploration were in progress. The elevations of the intersections obtained in the shore-line holes indicated that most of the ore lay between 200 and 600 feet below surface. It was decided that the 200 level of the old workings should be extended into the western ore area, where it would serve as a diamond drilling level and later as a service level above the greater part of the ore. At the same time, 900 level would be extended into the same area on an exploratory basis. A hoist, compressor, and other necessary plant were purchased and erected, the shaft was unwatered, and the levels rehabilitated. Driving northeasterly from the existing 200-level working began early in October. By early December, the first vertical holes to detail the ore were being drilled from the 200 level.

As soon as ice formed on the lake again that winter, diamond drilling from surface was resumed also. Ore intersections came rapidly from both the underground and surface drilling, and the structural picture began to appear.

By May, 1946 it was estimated that the drilling indicated at least 6,000,000 tons of ore averaging 0.181 oz. gold per ton, 1.5 per cent copper, and about 3 per cent zinc; and it appeared that another year's work would be required to round out the picture. By April 1947, the indicated ore totalled more than 9,000,000 tons grading 0.172 oz. gold and 0.92 oz. silver per ton, 1.52 per cent copper, and 2.69 per cent zinc. During 1947, 1948, and early 1949, the plants were being constructed and the mine prepared for production, which began in June, 1949.

This was a diamond drill development, in the sense that decisions affecting the scale of operations, design of mine and plant, and the ore dressing practices, were based almost entirely on information ob-

tained from diamond drilling. The mining experience so far has generally confirmed the estimates made from the drilling as to tonnage and grade of ore.

Mining Plants

Early in 1946, studies began to determine the location and capacity of a main operating shaft and the mining plant to service it. By that time the probability of a sizeable operation, and the possibility of later expansion, were apparent. It was decided to plan for an operation of 2,000 tons per day, and to provide for possible expansion in all decisions affecting location and design of the shaft and plant.

The topography near the ore zones is generally unfavourable for plant construction. Most areas not under water are low-lying and covered with clay overburden to depths well below the ground water level. The locations of the ore outlines, No. 2 shaft, and the milling plants are shown in Figure 1. Consideration was also given to locating the shaft and plants on the north shore of the lake, as indicated at the upper edge of this Figure. This location had the advantage that the mining and milling plants would be relatively close together. However, in 1946, the only indication of ore extending to depth was toward the southwest corner of the property, and the advantage of shaft location for probable future exploration outweighed that of a compact plant. The shaft site selected is on relatively high ground. This favourable area is limited, but it is adequate for the present mining plant and for expansion if required. Between this site and the milling plants there is a wide area of low-lying ground, less than 5 feet above lake level.

Selection of a site for the concentrator and its auxiliary buildings was more difficult. After the shaft decision had been made, it was obvious that they would have to be some distance from the shaft, but the question was, "How far?". The site on which they now stand was crossed by the branch line of the Canadian National railway, and was overlooked for some time. Eventually it was noted and, as the alternative site was even farther from the shaft, arrangements were made with the Railway Company to re-locate the line in its present position. Secondary crushing is done at No. 2 shaft, after which the ore is carried to the concentrator bins on belt conveyors totalling about 2,100 feet in length.

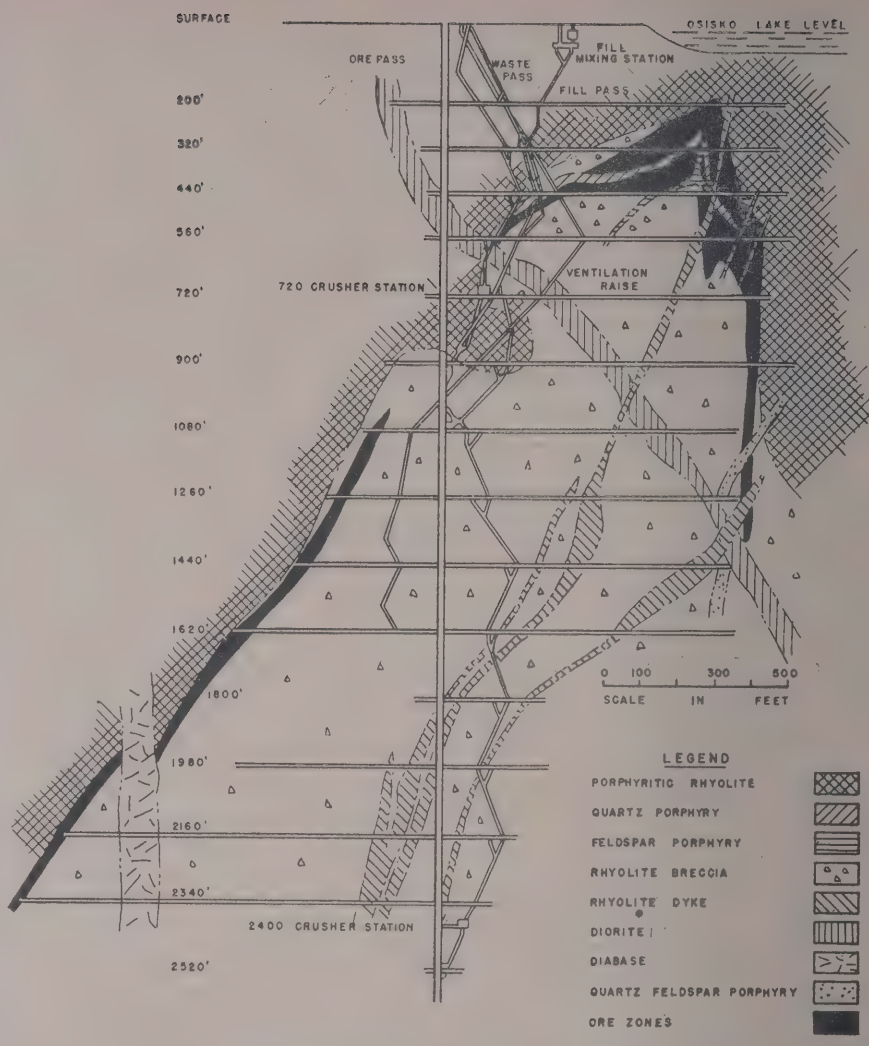

Figure 2.—Generalized west-east section through No. 2 shaft.

The headframe at No. 2 shaft is of steel, covered with corrugated aluminum siding. The shaft house at its base is constructed of tile on a structural steel frame, as are the other buildings on this site. The shaft house has the usual deck at yard level for handling timber and supplies into the mine. In addition, there is a lower deck below yard level, at which men enter and leave the main cage.

The general offices, the engineering offices, and the change house are combined in the large building northwest of the shaft. A closed passage, below yard level, connects this with the lower deck at the shaft head.

Two double-drum hoists are used. Drum size is 10 feet diameter by 7 feet. These drums will hold more than 3,400 feet of 1½-inch rope in two layers. The skip hoist is driven by two 600 h.p. motors, and its rope speed is 1,700 feet per minute. The cage hoist is driven by a 700 h.p. motor; its full rope speed is 1,500 feet per minute, but so far it has been operated at 1,000 f.p.m. Three air compressors deliver a total of 8,660 cubic feet per minute (2 at 3,330 and 1 at 2,000). This has been more than enough for all mining requirements to date.

All shops are consolidated in the large building northeast of the shaft. These include machine, plate, pipe, electric, and steel shops, and garage. These shops are adequate for all normal maintenance work. The proximity of Noranda and Rouyn, where there are several shops equipped to handle larger repair jobs, made it unnecessary to equip for complete self-maintenance. Subsurface tunnels connect all the main buildings on this site. All compressed air, water, and heating pipe lines, and power cables, pass through these tunnels and are readily accessible for maintenance or repair.

It may be noted on Figure 1 that there is space available to extend any of the buildings. There is also space behind the present hoist building to erect a larger skip hoist if required. Provision was made in the design of the headframe steel so

Figure 1.—General surface plan.

that it can be reinforced to permit the operation of such a hoist.

Ore Attitudes and Types

The Quemont orebodies occur in an area of acid volcanic rocks of Keewatin age. The formations consist of rhyolite breccia overlain by at least one thousand feet of porphyritic rhyolite folded into the shape of a dome with the flanks dipping to the north, east, and west. The south side is truncated by the steeply dipping Horne Creek fault. The deposits are sulphide replacements of the rhyolite breccia and occur beneath the dome of the impervious porphyritic rhyolite where flexures of the contact are favourable for ore deposition (Figures 2 and 3).

The ore may be divided into three classifications depending upon the degree of replacement and alteration of the breccia. These occur below the contact as massive sulphides, chloritized breccia with disseminated sulphides, and shattered breccia with sulphide stringers, usually in that order.

There are three main orebodies: the West, the East, and the Southwest. These separate deposits all lie below Osisko lake, and although the West orebody has a capping of porphyritic rhyolite, the East orebody extends through to ledge. The proximity of the lake poses a major problem in obtaining the highest possible ore extraction economically and safely.

The orebodies are intersected by numerous slips and several types of acid intrusives. In addition, the three ore types vary in specific gravity and hardness. These factors all add to the mining difficulty and a close control must be exercised for the optimum in grade, tonnage, and grindability.

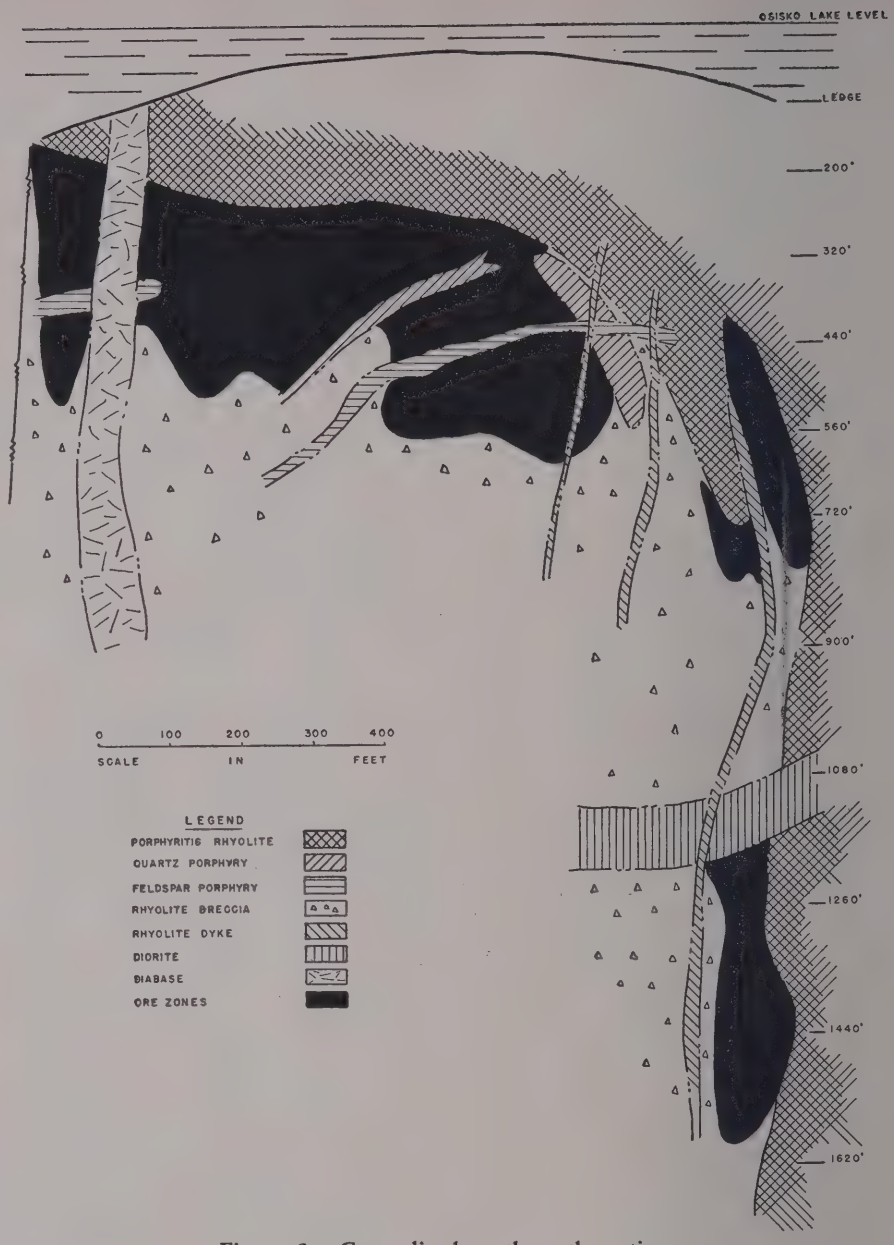

Figure 3.—Generalized south-north section.

No. 2 Shaft Design

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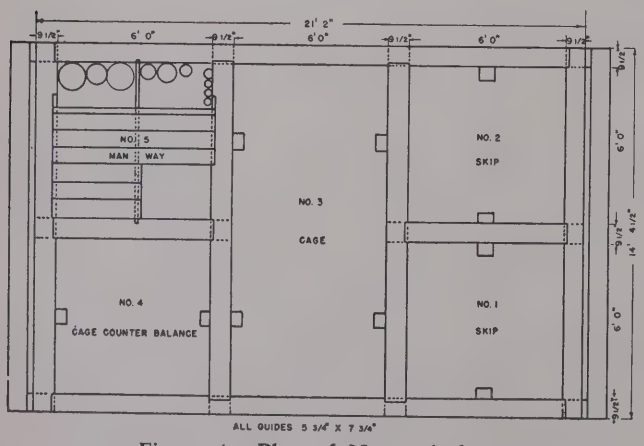


Figure 4.—Plan of No. 2 shaft set.

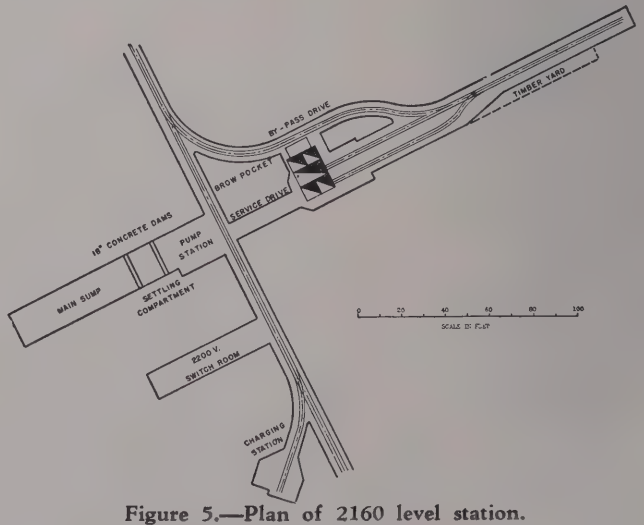


Figure 5.—Plan of 2160 level station.

2 shaft was to provide for a possible ultimate daily capacity of 5,000 tons from a depth of 3,000 feet. The design of the shaft set is shown in Figure 4. False end plates were used so that blocking could be placed at any point along the ends without distorting the No. 3 cage compartment.

The skips presently in service were designed for a minimum of 75 cubic feet working capacity and in practice the skip factor averages 6 tons. Skip dumps were designed to overturn a skip of 100 cubic feet, if the extra capacity is required. The large cage will hold 38 men, or 15 tons of material up to 12 feet in length on flat cars. A cage with an inside height exceeding 21 feet was placed in the counter-balance compartment and is used to service the mine with all material exceeding 12 feet in length. Skips and cages are equipped with pneumatic guide rollers. The service section of the manway compartment contains 10- and 6-inch air lines, a 3-inch water supply line, a 3-inch drain line, a 6-inch pump discharge line, a 16-inch ventilation pipe, and various electrical power lines.

Figure 5 illustrates the shaft station at the 2160 level. This is typical, except for the sump and pump room.

Pumping Stations

Pumping stations are located at the 1080 and 2160 levels, with a temporary installation at the 2520 level. The main sumps are as shown in Figure 5. They are 70 feet long, 16 feet wide, and 12 feet high. Two 18-inch concrete dams give a 59,000-gallon effective sump capacity and an 11,000-gallon settling compartment. The 1080 and 2160 pumping stations have identical static heads. Thus it was possible to purchase duplicate pumping equipment for each station, with attendant decrease in the stock of spare parts. Pump capacity is 300 gallons per minute at a total head of 1,125 feet.

Level Interval

The 200 and 900 levels were established and partially developed from No. 1 shaft. Diamond drilling of the block of ground between surface and the 900 level indicated approximately 85 per cent of the ore to be above the 560-foot elevation. To provide sufficient horizons for extraction of the required daily tonnage from the irregular ore sections indicated in the upper part of the block, a level interval of 120 feet was established down to the 560 level. This was followed by 160 feet to the 720 level leaving 180 feet to the 900 level. The interval of 180 feet has since been adopted for all levels below 900.

Ore and Waste Pass Systems

The initial underground crusher was located at the 720 level, with the loading pocket at the 900 level. Installation of a second crusher has been recently completed at the 2400 level, with a loading pocket at the 2520 level. Three surface bins serve as storage for crushed ore and waste. The upper ore and waste passes are provided with control chutes at all levels. The lower ore pass system, recently completed, extends to the 720 level. Waste passes below the 900 level will be driven as required for filling. Dumps are installed on all except the 200 level. Waste passes are provided with car loading chutes on all levels.

Ventilation

Ventilation is supplied by an 84-inch axial-flow pressure fan located on surface. It delivers 175,000 cubic feet per minute through a system of intake raises, presently extending to the 1620 level. These are equipped with regulator doors to control the supply of air for each level. The main level drives are equipped with an air-lock system of doors to isolate the ventilated section of the mine from the shaft area. An exhaust raise extends from the 320 level to surface.

Development

Level drives are 8 ft. by 8 ft. in cross-section. They are driven on line and grade by a two-man crew operating a 3-boom hydraulic jumbo. Rock drills are 3-inch drifters equipped with a 48-inch feed and using three changes of carbide-tipped steel up to 12½ feet long. Loading is done by a two-man crew using a *Model* 21 Eimco loader and standard haulage equipment. Advances up to eleven feet per round are made.

Raises are driven on line and grade on a single-shift basis. They vary in size from 5 ft. by 5 ft. to 6 ft. by 12 ft. Those steeper than 50 degrees are timbered with stull sets. Where a raise will be required for a long period as a serviceway, cribbed manway timber may be used. Rock drills are 2⅞-inch stopers using detachable forged-steel bits. Advances range from 6½ to 7½ feet per round.

Sub-drives are usually 7 ft. by 7 ft. in cross-section. Drilling may be done with two air-leg drills, completing one short round each shift, or with two 3½-inch drifters, completing one long round every two shifts. The broken rock is removed by scraping.

Stoping Methods

Due to the proximity of a major fault system and the presence of surface waters, it is essential that ground movement be kept to a minimum. To ensure this, it is necessary that complete filling closely follow ore extraction. Therefore, horizontal cut-and-fill stoping is used where the ore sections are irregular and of limited vertical extent. In sections of sufficient vertical extent and relatively stable ground, sub-level stoping, with adequate pillar support and followed by complete filling, is adopted. As extraction has progressed and a knowledge of the ground conditions has been gained, variations of the two initial methods have been introduced and utilized to advantage.

During the pre-production stage, areas of the West orebody on the 320, 440, and 560 levels were laid out and developed for both cut-and-fill and sub-level stoping. To keep the stockpile of pre-production ore to a minimum and still have sufficient tonnage immediately available at the start of milling, two sub-level stopes were prepared and blast-hole drilling started. Two additional sub-level stopes were partially prepared. Slashing of the cut-and-fill stope areas was deferred until milling had commenced. The success of this programme was demonstrated by the fact that the mill operated at capacity soon after the start of production, and within one month was handling over 1,800 tons daily, with only 700 tons being supplied from the stockpile.

Following the initial production period, development and stope preparation were extended to the East orebody above the 560 level, and more recently to the West orebody down to the 1620 level. Additional cut-and-fill and sub-level stopes are being prepared and mined in these areas.

Cut-and-Fill Stoping

Initial diamond drilling of the West orebody indicated an irregular flat-lying ore area of unknown continuity existing below the 200 level. To maintain adequate ground support until a plan could be selected for controlling the overlying lake water, it appeared that a large percentage of the ore would have to be left as pillars. Primary cut-and-

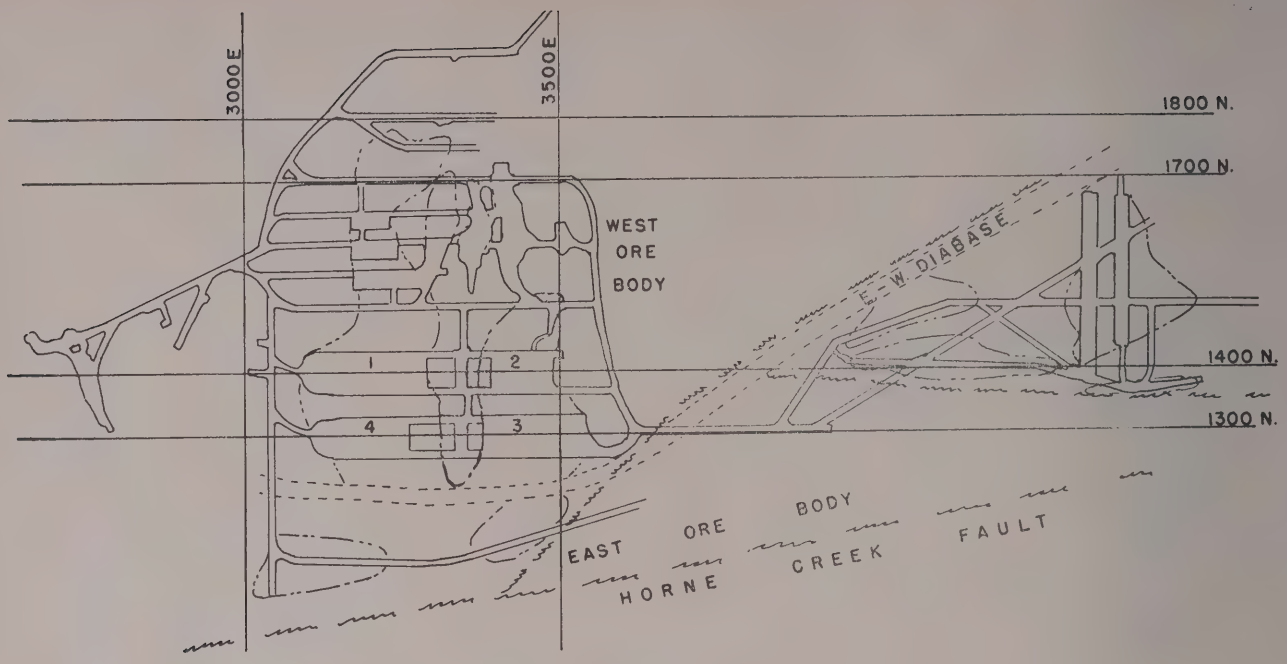


Figure 6.—Plan of 320 level.

fill stopes were laid out for mining widths of 30 feet, separated by 70-foot rib pillars. These were to be later subdivided into 30-foot secondary stopes and 20-foot pillars. This gave an overall mining plan of 30-foot stopes separated by 20-foot pillars.

Stope drives were then driven on 100-foot centres to service the primary stopes. This was followed by diamond drilling, which revealed that intrusive dykes occurring within the orebody were so large that they should be left and would serve as pillars. It was then apparent that, with the addition of these waste pillars, many of the secondary stopes could be mined simultaneously with the primary stopes, and north-south trending lenses of ore within this intrusive zone could be mined to assay walls. The stoping sequence was then revised for stope and pillar width of 30 feet and 20 feet respectively, these stopes to be mined simultaneously with the assay-wall stopes. Some assay-wall stopes have been considerably wider than 30 feet. In massive sulphide ore, these have been mined without timber support up to 60 feet wide. In other ore types, partial support is usually needed.

Experience with stopes of varying widths has indicated that both ground conditions and performance generally improve when widths are somewhat less than 30 feet. Based on this experience and also to assure more stable pillars, recent layouts provide 25-foot widths for both stopes and pillars.

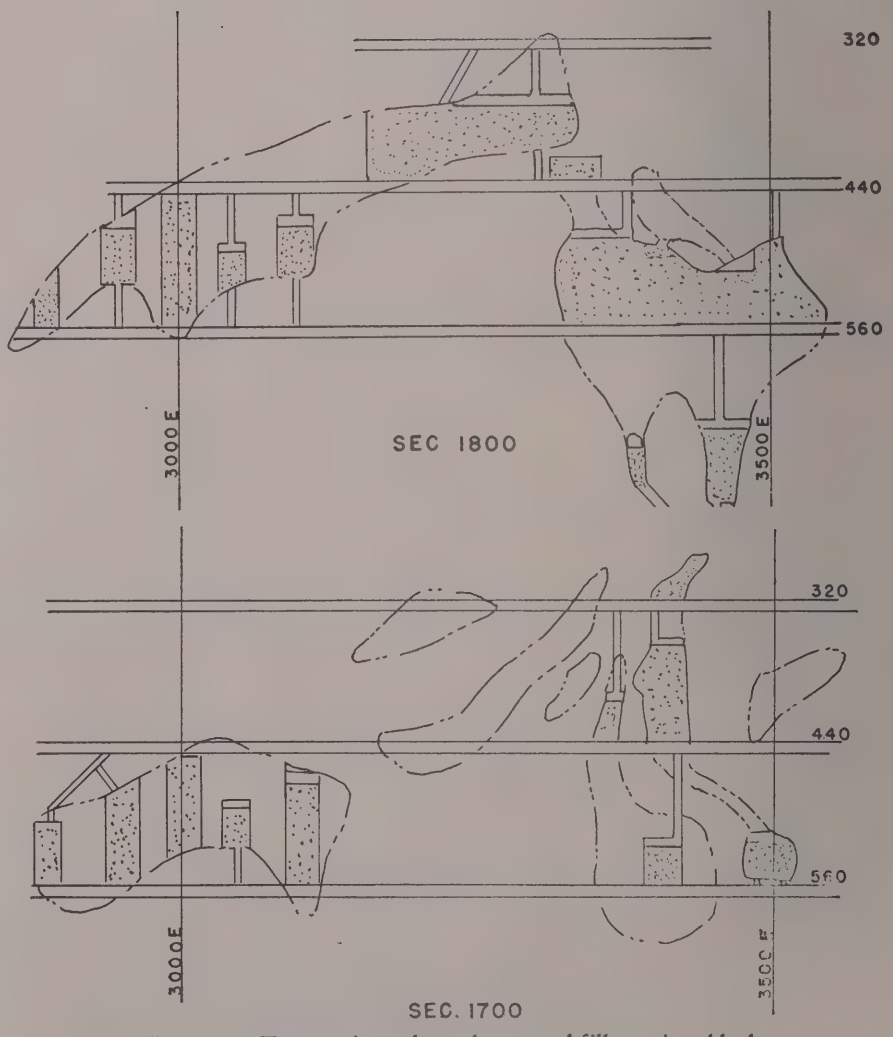


Figure 7.—Two sections through cut-and-fill stoping block.

Arc gate chute.

Stope breast.

Figure 6 is a plan of the 320 level, on which the areas of ore are greatest. These ore areas are outlined, and pillars and blocks not yet opened for stoping are shaded. Numbers 1 to 4 indicate sub-level stopes.

Figure 7 shows two sections through the cut-and-fill stoping areas.

Stopes are developed by 8 ft. by 8 ft. drives, and fill raises are driven through the proposed stoping sections. This is followed by slashing to pillar lines or assay limits at either rail elevation or 16 feet above rail, depending upon the ground conditions and the proposed crown pillar recovery methods for the stope immediately below. Slashing above the level is generally preferable in our opinion.

Sill timbering is standardized as far as practical, but, due to the irregularity of the stoping sections, detailed timber layouts showing millhole, manway, and fill dump locations are required for each stope. Depending upon haulage layout, the sill timber is installed either longitudinally the full length of the stope, or transversely across the stope at not more than 100-foot centres.

All chutes are designed with air-operated undercut arc gates (Figure 8). The one-piece detachable chute lip is easy to install and recover. Box-hole chutes are of similar design, and, where tonnage warrants, the sides and bottom inside the chute posts are concreted. Millholes are located depending on the sectional ore outlines, but not more than 100 feet apart.

Horizontal slices are taken 6 to 11 feet high, depending on ground conditions. Air-leg drills with carbide-tipped chisel steel are used for breasting. The back holes are drilled in the form of an arch, the centre hole being three feet above the side holes, and with the horizontal spacing limited to three feet (Figure 9). Grade marks for back elevations and pillar-line plugs are set by the surveyors. Millisecond delay caps are used for all blasting following the first lift above the timber.

The broken ore is removed by 3-drum, 30 h.p. electric scraper hoists equipped with 48-inch, hoe-type scrapers. All scraping is done from behind the breast, and, until breaking has been completed, only sufficient broken ore is removed to allow for the drilling set-up. Drum capacity is 210 feet of 5/8-inch rope. However, scraping efficiency drops rapidly at distances over 100

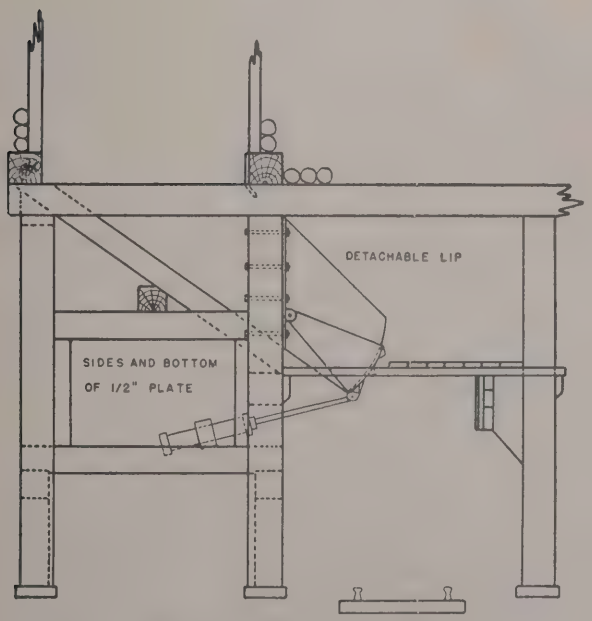


Figure 8.—Arc gate chute.

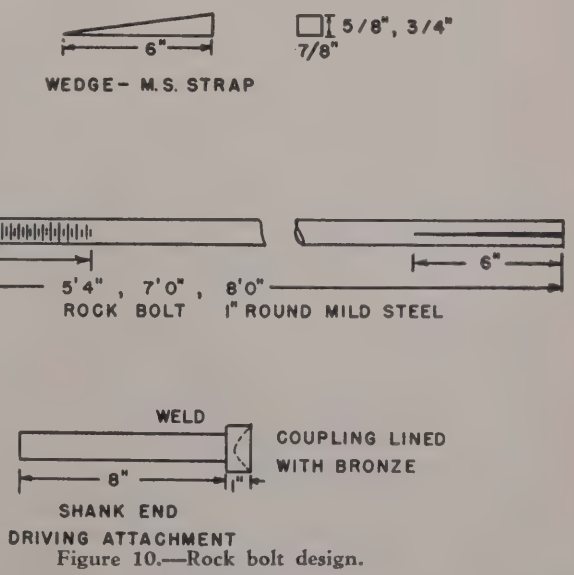


Figure 10.—Rock bolt design.

Preparing rock bolt.

Driving rock bolt.

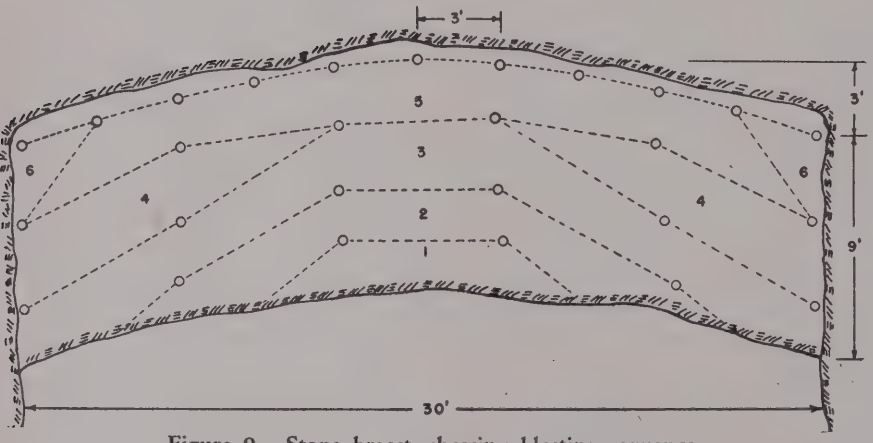


Figure 9.—Stope breast, showing blasting sequence.

fect. Performance varies with conditions and reaches 600 tons per shift. No grizzlies are used over the mill hole openings. Mill holes are left full and chutes not in use are sealed to prevent accidental drawing.

Fill is trammed to the fill raises by the regular haulage equipment. Air cylinders are used for dumping, or dumping ramps where tonnages justify the more costly installation. For spreading fill in the stopes, a light-weight, *V*-type box scraper is used. Fill grade marks are set by the surveyors.

Support of stope backs begins with the drilling pattern of closely spaced back holes along the 3-foot arch. Additional support to both back and walls is attained by the

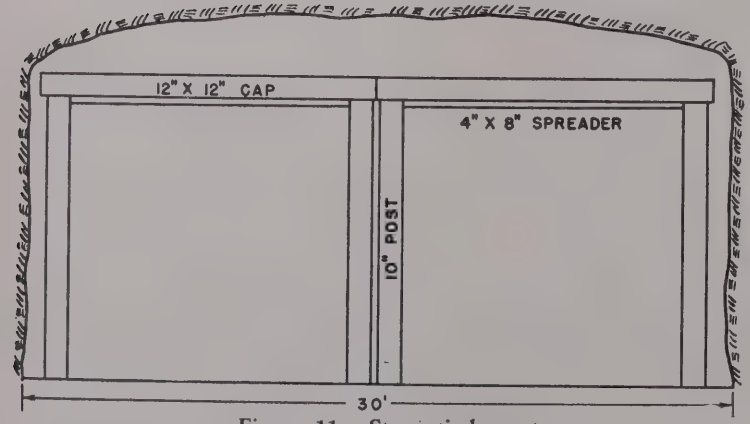


Figure 11.—Stope timber set.

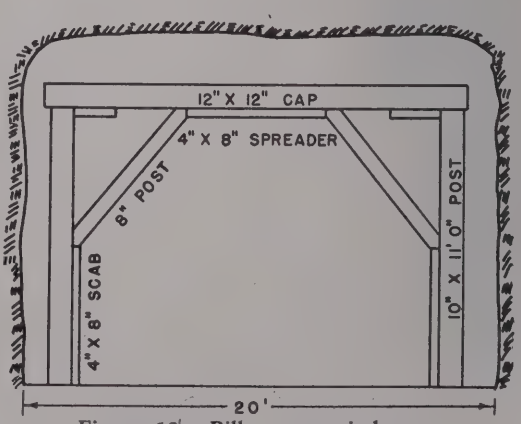


Figure 12.—Pillar stope timber set.

Timbered cut-and-fill stope.

Timbered pillar stope.

use of rock bolts. Where practical, bolting has the obvious advantage over timber support that the bolts can be installed immediately behind the breast before removal of the broken ore, and the full area left clear for scraping. The rock bolts are of split-wedge design (Figure 10), and are driven with either a stoper or air-leg drill, using the attachment shown.

In areas of weak and blocky ground, a procedure termed close timbering is used. The fill is levelled 6 feet below the stope back and a lift 8 feet high is mined. Timber sets, as illustrated in Figure 11, are placed at approximately 8-foot centres as the ore from each breast is scraped out. The mill hole is located within the timber set and the ore is scraped through the sets. Where ground conditions deteriorate to the extent that this method is inadequate, square-set timbering is used.

Pillar Mining

Wherever possible, pillar recovery is started at the bottom of the ore section. Sub-drives are driven through the centre of the pillar and slashed to the fill walls. If available, services from the adjacent mined-out stopes are used. Close timbering is usually required in these stopes (Figure 12).

Variations

Long Hole Benching of Cut-and-Fill Stopes. — This method is being used to advantage where a shallow elliptical, or saucer-shaped, ore lens extends below a level. It is illustrated by Figure 13, and is applicable only in strong ground as scraper operators are required to work within the stope opening during a large part of the recovery period. Following slashing, the back is arched from a height of 9 feet above rail at the pillar lines to 14 feet at the centre line. Rock bolts are placed on a pattern of 6 ft. by 8 ft., with additional bolts where required. Vertical bench holes with 4 ft. by 5 ft. spacing are drilled 2 feet below the ore limit and the bottom shaped for uniformity of pattern. Opening up around the slot proceeds with alternate blasting and drawing of the ore as long as all broken ore flows freely to the draw-off raise.

In the stope illustrated, the slot was opened to a width of 36 feet. The remainder of the bench was blasted in stages in the order shown, the number of holes in a blast being limited by the number of delays in the electric blasting cap series. After each blast, all ore that would flow by gravity was withdrawn to provide room for swell and to relieve the load against the face. The last stage was blasted with a vertical height of 24 feet of broken ore lying against the face. All holes were primed with heavily insulated

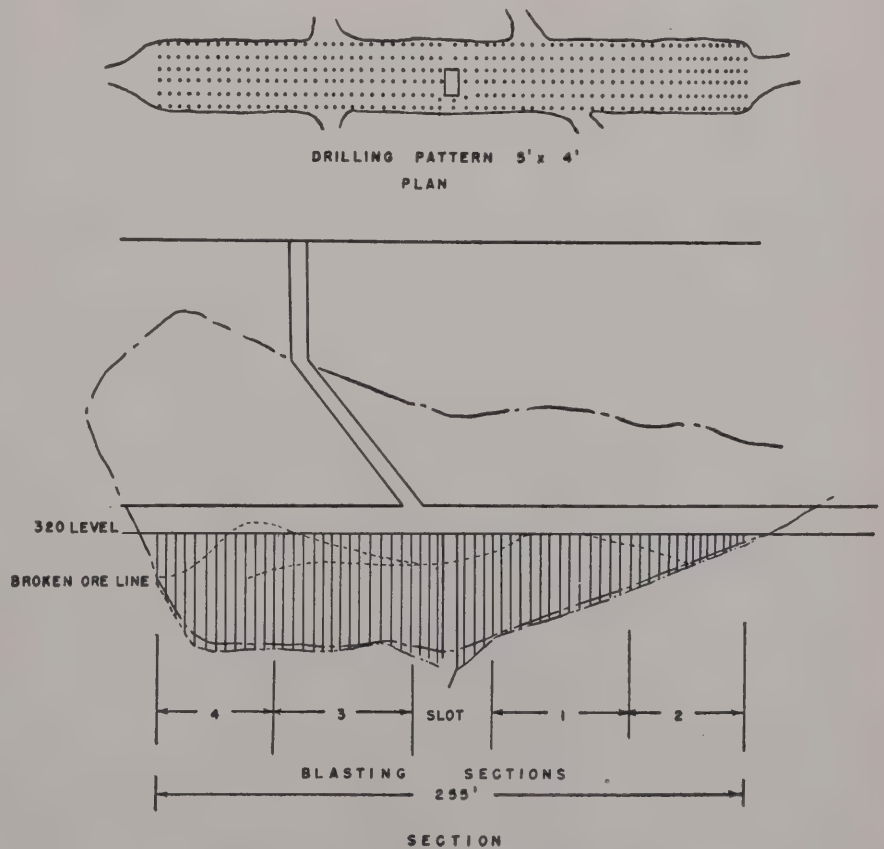


Figure 13.—Blast hole benching.

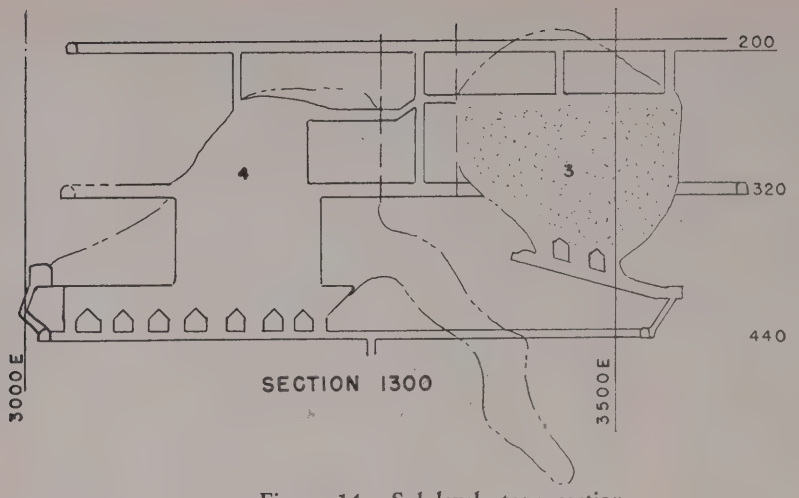


Figure 14.—Sub-level stope section.

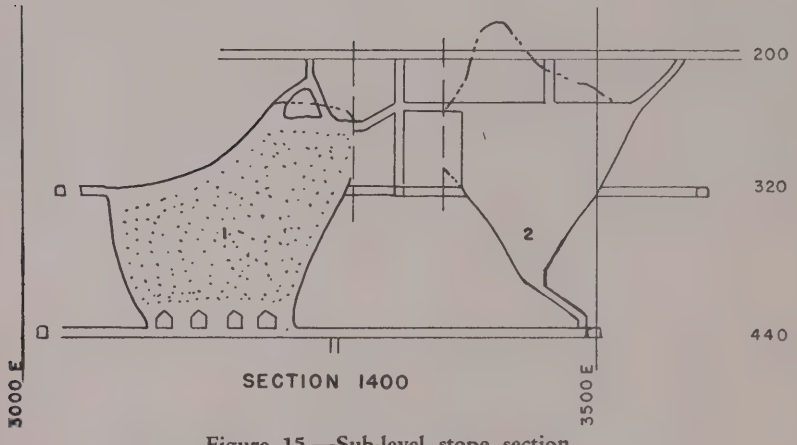
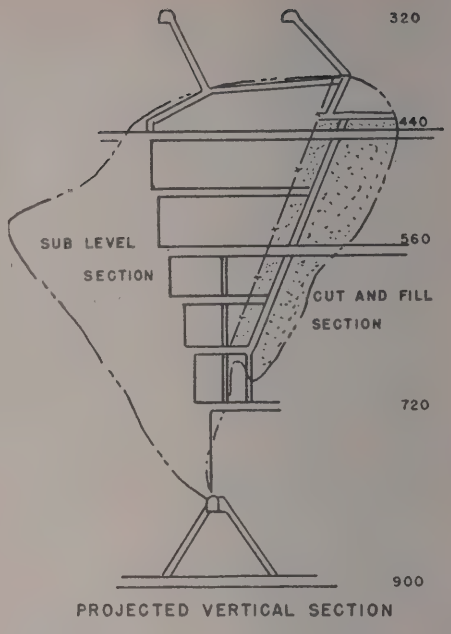


Figure 15.—Sub-level stope section.



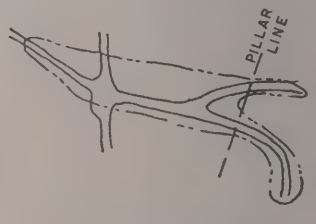


Figure 16.—Sub-level stoping, 21 orebody.

millisecond electric blasting caps. Pillar holes in each row had one period delay. To ensure against misfires, all holes primed with the same delay were connected by primacord. On completion of blasting, the stope back was given a final check and scraping of the broken ore proceeded. The walls were scaled and bolted as exposed. Fill was dumped from the level above and scraped to the required elevation.

Sub-Level Stoping

During the initial production period, four sub-level stopes (Figures 14 and 15) in the West orebody were prepared and operated to provide sufficient ore to maintain mill tonnage. As production from cut-and-fill mining increased, tonnage requirements from the original sub-level stopes decreased, making it possible to schedule a desirable sequence for their mining and filling. In general, Nos. 1 and 3 stopes were mined and filled, followed by No. 2 and No. 4 stopes. To maintain ground stability, crown pillars of massive ore were left in place below the porphyry capping. In No. 2 stope, the ore outline was suitable for a slide chute, adapted from the International Nickel design, and this has been a very efficient drawpoint.

Figure 16 shows plan and section of another area of the West orebody that is being mined by both sub-level and cut-and-fill methods. In this case, the narrower and more irregular areas at the east end of the lens are being mined by cut-and-fill in advance of the main sub-level section. A transverse pillar line, inclined at 70 degrees, separates the two stoping methods. The sub-level section will be finally retreated to the fill wall at the pillar line. Since the sub-level section is being mined to assay walls and is isolated sufficiently from neighbouring ore areas, filling may be delayed for some time.

Two-drum, 60 h.p. electric scraper hoists, equipped with 60-inch folding scrapers, are operated in the scraper drives. The tail sheave is either hung from a specially constructed reinforced *H*-beam concreted into the back of the drive, or is supported between two 12 in. by 12 in. fir posts hitched both top and bottom.

Sub-level stoping has produced approximately 35 per cent of the total mine tonnage to date. Its use at Quemont is limited and has been in conjunction with cut-and-fill mining, since the size, shape, and ground conditions of the orebodies do not lend themselves to complete extraction by this method. However, these stopes have proven particularly advantageous in providing a reserve, with which to balance tonnage and grade and to maintain relatively stable mill heads.

Filling

The filling material used is similar to that originally developed by Noranda Mines. Concentrator tailing, high in pyrrhotite, is mixed with slag, gravel, and waste rock. Oxidation of the pyrrhotite cements these sufficiently that pillars may be extracted below or alongside this fill.

Slag and gravel are dumped into a 2,000-ton bin located immediately below surface. The flow of fill from the bin to the fill pass is regulated by a vibrating Jeffery feeder, and at this point the concentrator tailing is added. Additional gravel and waste rock, in varying quantities, are dumped to the stopes along with the mixed fill. Fill requirements are estimated daily and monthly and the tonnage mixed is limited to immediate requirements.

Haulage and Hoisting

Haulage equipment consists of 60 cu. ft. Granby-type cars and 4-ton battery locomotives. A 6-ton Diesel locomotive is used for heavy haulage where the ventilation is adequate for its operation. In development work, where the hauls are short, 1½-ton locomotives are used. All track is either 40- or 60-pound rail, laid at 30-inch gauge and at 0.25 per cent grade favouring the ore haulage. Curves are generally of 40-foot radius, with 60-foot radius used where possible on main haulage drives.

Ore-pass dumps are located on the main level drives. These are equipped with air-operated, hinged ramps and hinged steel covers. The ramps and dump covers are controlled simultaneously by a 4-way valve.

The waste-pass dumps, control chutes, and loading chutes are located on the waste-pass drives, which extend off the main level drives. Grizzlies of 90-pound rail at 10-inch spacing are installed on the waste-pass dumps. Cars are dumped by a post-mounted air cylinder.

Mine ore is reduced to 4 inches by a 36 in. by 48 in. jaw crusher rated at 300 tons per hour. The flow to the crusher is controlled by a Ross chain feeder. Dust is collected by a multi-bag dust filter with a capacity of 3,000 cubic feet of air per minute.

Feed into the measuring pockets is controlled by air-operated, undercut arc gates. The chutes are equipped with control chains to prevent flooding. Feed into the skip is controlled by an air-operated, hinged door.

Development waste below the 1080 level is handled through shaft brow pockets. This rock is passed through a grizzly with 10-inch openings. Skips are loaded through air-operated, guillotine gates.

At the skip dump, the top part of the dumping apron is fixed while the lower part is mounted on a transfer car, which may be shifted to any one of three positions to direct rock to the proper bin.

Mechanical Maintenance

Mine mechanical operating maintenance, including the steel shop and drill repair shop, is under the direction of a Mine Mechanical Foreman. In addition to the shop employees, five day-shift mechanics, one afternoon-shift mechanic, two steel sharpeners, a pump man, and a crew on steel plate work, are employed underground.

Each mechanic is responsible for the equipment on his beat. Daily, or semi-weekly, schedules of inspection cover all mechanical equipment. To as great an extent as possible, repairs are made underground. Scheduled major overhauls are handled by the Plant Department in the main surface shops. The underground mechanics are responsible for the lubrication of all equipment except rock drills.

Carbide-tipped steel, used in the stopes, is sharpened in the working places, and the development steel is sharpened at the shaft stations. The employment of skilled sharpeners has aided in increasing the drilling life of carbide steel.

Electrification

Power (2,200-volt) is transferred to main underground switch rooms located on six levels. Four of these are equipped with 2,200-550-volt transformers, from which 550-volt power is distributed to switch rooms on other working levels. The switch rooms are equipped with a 550-volt feeder panel and a 5 k.va., 220-110-volt transformer.

Where scraper-hoist locations are centralized, main 550-volt distribution panels are suitably located on the level and are fed directly through armoured cable from the switch-room feeder panel. These then feed through the scraper-hoist control panel to the hoist. Where scraper hoists are isolated, the control panel is fed directly from the armoured cable. Distance from the armoured cable to the scraper hoist is kept to less than 500 feet to minimize voltage drop. The control panels for each 30 h.p. scraper hoist are located on the level near the stope manway, and trail cable is strung from here through the stope manway to the scraper hoist. In addition, each control panel is equipped with a 550-110-volt transformer to feed a 150-watt sealed beam light used in stoping operations. Scraper-drive hoist set-ups are identical except for size and permanency. In these cases, the distribution panel is eliminated, and the control panel is attached directly to the armoured cable in the immediate hoist area.

Station and haulageway lighting and traffic control lights are on the 110-volt line. Battery-charging stations are located on the main working levels. The 4-ton locomotive batteries are charged by Selenium or Rectox rectifiers, and 1½-ton locomotive batteries by Rectox rectifiers or motor-generator sets.

All electric blasting is done by the shift electrician on orders from the mine captain after all men have been checked out from the mine. An approved type 220-volt blasting switch is located in each level switch room. This is controlled by a double throw switch on the 220-110 volt lighting line. The plug-in to the blasting box is permanently shorted except when connected. Neoprene cabtire cable is strung throughout the level, with receptacles located near each working place. The plug-in for these receptacles is also provided with a shorting device and the cabtire strung into the working place. When preparing for a blast, the electric blasting caps are connected by lead wire to the cabtire, and the plug is un-shorted and connected to the receptacle on the main blasting line.

Mine Organization

The mining staff is organized in three sections, Operating, Engineering, and Geology, responsible to the Mine Superintendent.

1. — *Operating*

This section is under the Assistant Mine Superintendent. The mine is in two divisions, each of which is in charge of a mine captain. The mine mechanical foreman is in charge of underground mechanical equipment. Underground electricians work under an electrical foreman who is responsible to the chief electrician.

2. — *Engineering*

This section is divided into three sub-sections, as follows, under the Chief Mine Engineer:

(*A*) *Production.* — This engineering sub-section was primarily organized for mine planning and layouts. Due to the irregularities of the ore outlines, extensive planning is required before an ore block can be developed for production. A flexible, scheduled rate of extraction is established for each orebody over a several-year period. Individual orebodies are divided into blocks and general mining layouts and schedules are drawn up for each block. From

these general layouts, stope preparation is scheduled. Detailed layouts are then prepared and made available to the operating staff. To maintain required tonnage and grade in line with the best operating schedule, estimates of tonnage and grade from each working place are made for monthly, quarterly, and yearly periods. This sub-section also prepares estimates of future requirements of most mine supplies, such as mechanical equipment, timber, cement, rails, pipe, etc.

(*B*) *Efficiency Engineering.* — This sub-section handles incentive bonus, mine ventilation, control and distribution of backfill, and research and testing problems that arise in the use of new operating equipment.

(*C*) *Surveying.* — This sub-section is in charge of a senior surveyor and office engineer. The latter is in charge of engineering maps, calculations, survey reports, and filing. The surveyors handle all underground and surface surveying, calculation, mark ups, etc., and work in close co-operation with the operating staff.

3. — *Geology*

This section is headed by the mine geologist, and is divided into geology and sampling sub-sections. The geologists direct exploration driving and diamond drilling, and handle geological mapping, ore reserves, etc. Sampling results are tabulated and stope assay plans are distributed daily to the mine staff.

Acknowledgments

The authors thank the President and Directors of the Company for permission to prepare and present this paper.

Many members of the mining staff contributed to its preparation, and their help is appreciated. Mr. C. S. Boland, until recently General Superintendent, gave valuable advice and assistance. In his earlier position of Mine Superintendent he was generally responsible for planning the mine and preparing it for production.

Our Distinguished Patron's Valedictory

TEXT OF VISCOUNT ALEXANDER'S ADDRESS DELIVERED AT THE ANNUAL DINNER OF THE INSTITUTE, OTTAWA, JANUARY 22ND, 1952, ON THE OCCASION OF HIS LAST PUBLIC APPEARANCE AS GOVERNOR-GENERAL OF CANADA

(*Transactions, Volume LV*, 1952, pp. 59-60)

I AM VERY HAPPY to be here this evening to enjoy the good fellowship and companionship of such a distinguished body as the members of the Canadian Institute of Mining and Metallurgy.

I had the pleasure of addressing you soon after I came to Canada — that was five years ago almost to the day.

Tonight I address you just before I leave Canada and I must admit that the occasion holds a note of sadness for me. My wife and I have been very happy here in this country amongst you. My children have grown up as young Canadians and are frankly very unhappy at the thought of leaving Canada, which has become our second home.

However, I am grateful that my life has been so ordered that I have been able to witness something of the growth of this country, both as a nation and as a rapidly developing industrial power.

FATE KIND TO CANADA

If fate has been kind to some people, we must surely recognize that she has been favourably disposed to the Canadian people.

During the comparatively short time that I have been here, we have witnessed the discovery of two major sources of wealth, namely, the oilfields of Alberta and the great ore deposits of Ungava. And this is in addition to the mineral wealth which is already in full production — many of these latter mines I have visited.

I recall with special pleasure my visit to the International Nickel mines at Sudbury — and later my visit to Yellowknife, when Dr. Bateman was my host and when Dr. Jolliffe explained to me the significance of a gold-bearing outcrop which one knelt on. I must admit that at the time I was not particularly impressed because I could see no chunks of gold, as I had expected. But I had a suspicion that Dr. Jolliffe knew more about it than I did — and furthermore, I ought to have been prepared, because I had already read a little book sent to me by its author, Mr. Cummins of Calgary, entitled *Amateur Prospecting*, in which he gives some wise advice in saying "all is not gold which glitters."

I think this applied to the pioneers who discovered Flin Flon.

INTRIGUED BY NAME

I had always been intrigued by that name and it wasn't until I visited Flin Flon that I learned from the late Tom Creighton the origin of the name. It was he who told me that they called it Flin Flon from a novel they found in a trapper's cabin, on the site of their discovery. The novel was all about an American professor by the name of Josiah Flintabbity Flonation, who went down into the bowels of the earth and discovered a world where everything was made of gold except the king's crown, which was made of tin. Tom Creighton thought that he had struck gold but it turned out to be copper.

It is somewhat amusing to think that the story in that novel, written in 1905, poked fun at the king's crown, which was made of tin. A lot of people today would like to have more of that desirable metal.

Of the great mining industries of Canada, I have only mentioned three, but except for a very few, I think I have seen them all.

As a layman in these matters, I can say nothing which is of any use to you, but my first-hand experiences have been of inestimable value to me in assessing the wealth and potential wealth of our country.

VAST MINING DEVELOPMENT

What impresses me is the vast development which has been made in the mining industry since I first came to Canada six years ago.

In 1946, the value of Canada's mineral production was some $500,000,000.

In 1950, four years later, the annual value had practically doubled.

In the fifth year, that is in 1951, the annual total exceeded $1,228,000,000.

I will not burden you with statistics, but I cannot refrain from the pleasure of stating that during the past five years, such valuable commodities as the following have made these remarkable advances in dollar values: aluminum has trebled from $60,000,000 to close to $180,000,000; asbestos has trebled; cement has doubled; copper has trebled; gold up 60 per cent; iron ore has quadrupled; lead over doubled; nickel has risen from $45,000,000 to $150,000,000; petroleum from $15,000,000 to $121,000,000; zinc from $37,000,000 to $133,000,000.

Truly remarkable figures!

RECALLS ARVIDA VISIT

I remember with great pleasure my visit to the great aluminum plant at Arvida. Here again we have a name made up of the first two letters of the names of one of the great pioneers in this development — Arthur Vining Davis.

In 1946, the dollar value of Canadian prime aluminum output was $60,000,000.

In 1951, the dollar value of Canadian prime aluminum output was $176,000,000.

Expansion in the still young aluminum industry has resulted in a marked increase in production cost efficiency. Canadian selling price in 1939 was 22 cents a pound. In 1951, in spite of the marked inflationary trend of both raw materials and labour, the selling price stands at 18 cents a pound or 18 per cent below pre-war level. Whereas prices of other common non-ferrous metals have advanced in cases as much as 300 per cent beyond their pre-war levels, aluminum alone is selling for less than it was prior to World War II.

Throughout its short history,

Canada's aluminum industry has been progressive in expanding to meet the demand of the world markets which it serves. Second only to the United States, in total output, Canada is the world's leading exporter of aluminum.

Aluminum Expansion

In anticipation of increasing consumption of aluminum for commercial and national defence applications, a vast programme of expansion was started two years ago. In the Province of Quebec alone, the year 1951 saw one new pot-line completed at Isle Maligne, and another stand-by plant put into operation at Beauharnois. A further new pot-line is due to start production at Isle Maligne in 1952.

Accompanying these plants are hydro-electric developments totalling 400,000 horsepower and costing about $100,000,000. A further expansion was initiated in British Columbia during 1951 which will shortly add a minimum of 20 per cent to the present total of Canadian output, for an expenditure of $166,000,000. Facilities, however, are provided to allow a potential increase in output which could double the existing production of aluminum, for a total investment of $550,000,000.

Canada is accordingly prepared to maintain her position as an efficient and major supplier of aluminum to the world whenever the increase in demand may so dictate the need for such additional expansion.

Wealth Has Obligations

All this means wealth to Canada and its people, with increasing opportunities for the future. But it won't be all money for jam — wealth has its obligations. The rich cannot enjoy their wealth selfishly — wealth brings responsibilities together with all its troubles, and Canada will find, as she grows in richness and prosperity, that she will be drawn more and more into the world's problems and have to shoulder greater responsibilities than are as yet realized.

Knowing the Canadian people as I do, I have no fears that they will shy away from the obligations which the future will impose upon them — or that they will be reluctant to shoulder the responsibilities which the world will expect them to carry. On the contrary, Canada, with only 14,000,000 people, has shown her readiness and willingness in world affairs. If fate has been kind to Canada in nature's gifts, fate could not have chosen a finer people upon whom to shower her favours.

As I see it, much of the world's difficulties today lie in an uneven distribution of nature's wealth in relation to population.

This of course, is not entirely true. Some of the world's largest populations are living in poverty because they have not yet developed their existing natural resources, but there are certain nations who have not got the resources to develop. I cannot think of a better example in this respect than Italy, which has neither coal, iron, nor oil and a very limited agricultural economy to sustain a population of 40,000,000.

Problem Facing Canada

In this respect, therefore, the problem which faces you is to make the best use of your wealth by exporting what you don't need for your own domestic use — or for manufactured goods which you can also export.

What you have here in Canada, the modern age demands in ever increasing degree; not only for civil needs but for the defence organization under the United Nations and *NATO*.

I have touched on something of your vast natural existing and potential resources, but I have said nothing about the personnel who produce them.

It is obvious that for a long time to come there will be an ever increasing demand for trained engineers, expert geologists, mineralogists, and technicians of all types. And in this field alone, there is a wonderful future for your men with enterprise and initiative and, above all, the will to learn and work.

Lauds Institute

In this wise, I should like to offer my congratulations to your Institute, which is indeed a major driving force in the remarkable development of the mining industry of Canada.

I feel quite sure that one reason the mineral industry has progressed so successfully during the last years has been because the Canadian Institute of Mining and Metallurgy has lived up to the prime objective of its charter — namely: "To promote the arts and sciences connected with the economical production of valuable minerals and metals".

You may well be proud of the work your Institute has done for Canada over the last 50 odd years. May I close by extending to you my best wishes for further and greater success in the next half century which lies ahead of us.

Operating Cost Budgets for Mines

By LEONARD B. KELLOGG *

(*Annual Western Meeting, Calgary, Alta., October,* 1951)

(*Transactions, Volume LV,* 1952, pp. 61-68)

Introduction

THE MINE MANAGER or Superintendent is a man faced with a difficult problem. He must direct the activities of his mine in such a manner that, over a period of time, the greatest possible profit will be realized. In this respect, a mine is no different from any other business organized for profit. What makes the mine manager's task so difficult then? The profit of a mine is determined by — among other things — the grade of ore, the market price of the metal or metals produced, and the cost to operate the mine. The mine manager cannot do much about the first two of these, but he has considerable control over the operating cost. This cost will depend in large measure on the nature of the programme on which he plans to operate his mine. This operating plan may be broad or specific, on a long-range basis or on a month to month basis. The development of the plan, however, is made very difficult by the existence of so many factors which must be considered. These may be (1) factors requiring decisions based on the best judgment and interpretation of engineering and geological information available, or (2) items which must be considered but which are largely non-controllable. To be more specific, examples of such items would be:

(1).—(*a*) The amount and type of development work to be done in relation to amount of ore to be mined.

(*b*) Mining methods to be followed and type of equipment to be used.

(*c*) Amount of exploration diamond drilling to be done if the orebody is not clearly defined.

(2).—(*a*) Rock type and degree of hardness.

(*b*) Width, depth, and pitch of veins.

(*c*) Amount of work requiring timbering.

(*d*) Faults encountered.

(*e*) Water and gas conditions met with.

This is only a partial list. In some cases, there will be many more factors requiring consideration. Is it any wonder, then, that the mine superintendent gets grey hair in trying to determine what the proper relationship of all these items should be? Having decided finally on the proper course of action he feels should be followed in operating his mine, he proceeds to adhere to it as closely as possible.

His next concern will be to operate against this plan at the lowest possible cost. If the current cost appears unduly high, the cost must be examined further.

With some companies, it is the practice to compare current operating costs with similar costs for the preceding month or for the same month of the previous year. This is often difficult and the result may be quite misleading, as the costs with which the comparison is made may have been unusual for some reason and it then becomes necessary to analyse two different sets of costs in order to find the true explanation of any difference there may be between them. Even worse, a trouble spot may be overlooked entirely because costs in the previous year may have been so high that current costs, even though higher than they should be, are considerably lower and thus appear in a favourable light. There are usually other factors tending to make such a comparison difficult, such as differences in wage rates and in the level of activity in the two periods.

Since comparisons of this kind do not give the required information easily or accurately, other companies have adopted the practice of comparing costs against a fixed budget. Costs are compared each month with a fixed amount, regardless of the volume of production attained or of fluctuations in other operating conditions. This, of course, is very misleading where actual conditions differ appreciably from those obtaining at the time the fixed budget for the year was established.

Because of the shortcomings of the fixed budget type of cost comparison, many industrial concerns have found the answer to their cost analysis problems by using a 'flexible budgetary control' plan. Though this plan has proven itself in industry for many years, it is only recently that it has been adopted by some of our mining companies. Because of the complexity of mining operations and of the success of this type of cost analysis in presenting the costs in their proper relationship to one another, its future in this field holds much promise.

Flexible Budget Control

Flexible budgetary control can be defined as the control of costs, and the co-ordination of all phases of the operation exercised by management, through the comparison of actual costs with those of a carefully predetermined plan of operation translated into dollars.

There are three very important things a mine manager needs to know about his costs each month:

(1) What has been the effect on costs of the current level of activity in the mine?

(2) What has been the effect on costs of a change in plans in the amount of the various types of work to be done?

(3) What has been the effect on costs of performance being off standard for the work actually done?

These questions are all answered under a flexible budgetary control system and the answers are pre-

*Stevenson & Kellogg, Limited, Vancouver, B.C.

sented as a matter of routine on the monthly cost statements. How this is done will be discussed after a little more background information.

In preparing a flexible budget, costs of the operations are collected by grouping costs of related operations into 'cost centres'. Cost centres are selected to conform with the responsibilities of the various supervisors in the organization and with regard to phases of the operation which are of like nature.

Within each cost centre, costs are further classified as fixed, variable, mixed, or irregular. Fixed costs (sometimes called 'period' costs because they are uniform for each period) are those costs which should remain constant over the budget period. Variable costs are those which should vary in direct proportion with the activity of the cost centre. Mixed costs are those that include a fixed element and a variable element. Irregular costs vary in some irregular manner.

This classification and analysis of costs between fixed, variable, mixed, and irregular is what enables us to compute the various cost relationships and is what characterizes the flexible feature of budgetary control. By 'flexible' is meant that the budget allowances, against which the actual costs are compared, are adjusted to take into consideration current level of activity, seasonal variations, and any other variations of an irregular nature that can be predicted.

Flexible budgetary control places emphasis on planning ahead. This is the first prerequisite of a good budget. Experience with the system where it is already in operation in mines of varying size and degree of complexity has indicated that it is possible to forecast mining operations for a year ahead with sufficient accuracy to allow a good budget to be set, even though in some cases the nature of the orebody may be very erratic. This type of cost control is often referred to as an engineering approach to cost analysis, in that cost forecasts and comparisons are based on an engineering plan of operation translated into dollars.

The plan of operations for the year is determined and stated in terms of each cost centre's unit of activity, such as feet of advance of development work, tons of ore to be stoped, and feet of diamond drilling. These figures are normally divided by twelve and the result is called the 'normal activity' for a normal month. This is the activity at which the standard unit cost of each cost centre is determined and is the basis of all comparisons.

It is unlikely, however, that the actual activity attained in any month will correspond exactly to the 'normal' activity. Since some of the costs are fixed, the unit costs, at levels of activity greater than normal, will be lower than the budgeted cost because of the greater number of units over which to spread these fixed costs. Correspondingly, at levels of activity less than normal the unit costs will be higher than the budgeted cost because of the lesser number of units over which to spread the fixed costs. This difference in costs is called the 'volume effect variance'.

It is also unlikely that the actual work done throughout the mine in any month will be at the same level and in the same proportion as was anticipated to be averaged over the year. Since part of the work may be low-cost work and account for a relatively small number of dollars in the overall budget, it follows that, if more, in proportion to the total forecast, of this type of work is done during the month, the overall unit cost will be lower than the standard cost. This difference in unit cost is called the 'mixture variance' and results solely from the various classes of work actually done differing, in proportion to the total, from what was planned to be done.

Having isolated the volume variance, which shows the effect on costs of the amount of work done, and the mixture effect variance, which is a cost measurement of the class of work done, management is interested in knowing whether or not the costs are in line for the work which was actually done. In order to know this, the actual costs charged to each account must be compared with a budget allowance which has been adjusted to reflect the actual activity of the period. The costs have been classified in the budget as fixed, variable, mixed, or irregular. In any month, then, the current budget allowance for fixed items will be the same as in any other month. For variable items, it is calculated by adjusting the normal allowance in the budget to the activity of the period. The allowance for items classified as mixed is determined by adding the fixed portion to the variable portion adjusted to the activity of the period. The allowance for items classified as irregular is calculated as detailed in the budget for the period. The difference between the actual cost and the current budget allowance is known as the 'operating cost variance'.

The operating cost variance is often broken down further for certain key items of supplies or labour into the portions that have been caused by variations from standard in price or rate and in quantities used. These are referred to as 'price variance' and 'usage variance'. They serve to isolate further the controllable from the non-controllable type of expense.

In order to demonstrate more forcefully how these cost concepts are put into actual practice, a sample budget for a typical medium-sized mine has been prepared together with sample operating cost statements for a representative month. The example follows this preliminary discussion. The complex cost relationships existing in a mine have been sorted out and analyzed, with examples of each of the situations discussed.

It has been usual to regard budgets only as a useful aid in the control of operating expenses, but they should be looked at in the broader sense that they can, as well, be a very valuable basis for profit planning. They tend to put the house in order, as it were, by compelling management to study and to plan for the most economical use of labour and material, with an ever vigilent eye on expense. It is the only means for predetermining when and to what extent financing will be necessary, and managements which have developed a well-ordered budget plan, and which operate accordingly, will find greater favour from their bankers and boards of directors than those which have failed to do so.

To insure the success of a budgetary control plan, operating men must have sufficient part in the preparation of the budget that they will understand the significance of subsequent reports of actual results as compared with it. It is the usual practice, therefore, to have the operators prepare their own budgets, assisted and guided by someone trained in the mechanics of the system, and then forward their estimates through the normal channels for approval.

During the discussion so far, you have no doubt been thinking of the cost situations and problems dealt with in the light of your own mining, milling, or metallurgical operation. Perhaps you have even considered the possibility of operating under a budgetary control plan. If so, the next questions that would have to be considered are: How would you go about adopting the plan? What would be its cost? How long would

it take to perfect it? What is its upkeep cost? Let us take them in order.

Before proceeding to put the plan in force, another point must be decided. Should we try to do it ourselves or should we obtain outside assistance, qualified and experienced in this type of work? Even though it may be felt that there is some person in the organization with sufficient time and knowledge to do the work, our experience has indicated there are distinct advantages in having it directed by someone with an outside point of view and with the experience accumulated through many years of familiarity with the system. Confusion will be minimized, the programme will not bog down because of indecision as to the next step to take, and in the long run the cost in time and money expended will be less.

The cost and time to introduce and perfect the system are both dependent on the size of mine involved, the condition of the cost system in use, and what records and standards of performance are in existence. Although it is difficult to assess in dollars the savings and benefits accruing from the use of a flexible budgetary control system, the concensus is that they far exceed the initial cost involved in its adoption.

During the period of working out the details of the system, one or more men, depending on the wish of the management, are usually assigned, full-time, to assist in the work and so become thoroughly indoctrinated in the principles and mechanics of operating the system. They will then be in the position that they, themselves, can make any revisions or additions that might become necessary in the course of time.

The budget is usually reviewed and revised as necessary once a year. This task must be the responsibility of one man who is familiar with the budgetary techniques. Other than this annual revision there should be no extra labour required to operate the system. In fact, the time that executives normally spend in trying to analyse and interpret the costs under the usual type of cost presentation is saved, since they are presented to him in an orderly fashion and he can direct his attention wholly to the critical points and not to a *search* for the critical points.

Though the foregoing discussion has dealt chiefly with budgets for mining costs, it should be mentioned that flexible budgetary control has been applied to milling operations as well. Experience indicates that, in its operation and in the results shown, it is equally as effective for milling plants as for mines. One of the things brought out by the use of flexible budgets, and one which mill operators have found useful, is the cost effect of variations in the price and usage of reagents and other major items of supplies. Also, these budgets have shown where there is a need for, or alternatively evidence of, a definite plan of preventive and routine maintenance work which, over a period of time, results in lower costs to operate the mill and in less 'down' time.

It is hoped that this discussion of costs, and the example which follows, will create the desire for further information concerning the application of flexible budgetary control to mining and milling.

Sample Budget and Operating Cost Statements S. & K. Mine

Flexible budgetary control principles and procedures are applied in this example to prepare a hypothetical budget for a typical medium-sized mine. Following this, sample operating cost statements for a representative month are discussed to illustrate how use is made of the budget in analyzing current costs for better control.

In the example, for the sake of simplicity, only the major variables will be considered. Such items as stope preparation work, ore obtained from development, and diamond drilling, all of which would be taken into consideration when building up a budget in actual practice, will not be discussed.

Normal Activities

Forecast activities as shown in Table I will be assumed.

Budget Construction

A budget for a cost centre can be built-up in either of two ways, depending on what is considered to be best for the cost centre in question. Thus, for stoping and development, the budget is usually built up according to class of expense (labour, supplies, etc.), and for cost centres which include a number of diverse types of work of a more general nature, the costs are budgeted according to the type of work (air and water lines, shaft maintenance, underground clean-up, etc.). In the example, one of each type will be considered.

A budget detail sheet is prepared for each code or each class of expense of the Cost Centre. On this sheet is shown all the pertinent cost information, such as a description of the items to be charged to the code, basic standards data for performance, usage or prices, the build-up of the normal budget allowance, and how the budget allowance for any particular month will be calculated.

The budget detail sheets in the cost centres are summarized, and the summaries are in turn summarized on a budget summary sheet covering all cost centres included in the costs of the mine.

Stoping Budget

Figure 1 is an example of a budget detail sheet for stoping labour. On it there are three major headings: *Standard Data, Normal Budget Allowance,* and *Current Budget Allowance.* Under the first heading, since this is a labour account, there is a further breakdown according to contract labour and other labour, each of which is budgeted according to usage in terms of man-shifts and to price in terms of $/man-shift. This information, together with the forecast tonnages by stope sizes, is all that is needed to arrive at the normal budget allowance for stoping labour. How this allowance is arrived at is shown in tabular form under *Normal Budget Allowance.* Since the tonnages produced each month will differ from the normal tonnages

Table I

Stoping: 300 tons/day, 25.5 days/average month = 7,650 tons/month

Stopes: 0 to 5,000 tons.....2,000 tons
5,000 tons and over.....5,650 "

Development — Feet Advance

Rock Type	Drifts and X-Cuts		Raises		Total	
	Year	Month	Year	Month	Year	Month
A.....	4,800	400	840	70	5,650	470
B.....	2,460	205	480	40	2,940	245
Total....	7,260	605	1,320	110	8,580	715

S. & K. MINE

BUDGET DATA FOR 19

Standard Period Month	Code Name Operating Labor	Cost Center Stoping
Date	Standard Unit Tons Ore Produced	Normal Budget $ 4,766

STANDARD DATA

Contract Labor

Usage - 0 - 5,000 tons size - 33.3 tons/MS, or 3.00 MS/100 tons
5,000 tons & over - 40.0 tons/MS, or 2.50 MS/100 tons

Price - Stope Miners @ $11.00/MS
Contract $ earned $ 3.00/MS

Day Pay Labor	Usage		Price
Timberman	4 MS/day	@	$11.50
Powderman	1 MS/day	@	10.50
Muckers	2 MS/day	@	10.50
Total	7 MS/day	@	$11.07

(or 7 MS ÷ 300 tons/day = 2.33 MS/100 tons @ $11.07/MS)

NORMAL BUDGET ALLOWANCE

Contract Labor	Stope Size	MS/Ton	Tons	Total MS	$/MS	Total $
Stoping	0 - 5,000	.030	2000	60.00		
	5,000 & over	.025	5650	141.25		
Contract $ earned				201.25	11.00	$ 2,214
				201.25	3.00	604
Total Contract Labor						$ 2,818
Day Pay Labor		.023	7650	175.95	11.07	1,948
Total Normal Budget Allowance						$ 4,766

CURRENT BUDGET ALLOWANCE

$.4200 x No. tons produced from 0 - 5,000 ton stopes
+ .3500 x No. tons produced from 5,000 tons & over stopes
+ .2546 x No. tons produced from all stopes

Figure 1.—Detail sheet for stoping labour.

S. & K. MINE

BUDGET DATA FOR 19

Standard Period Month	Code Name Operating Supplies	Cost Center Stoping
Date	Standard Unit Tons Ore Produced	Normal Budget $ 2,213

STANDARD DATA

Usage - 0 - 5,000 ton size stopes 2.00 lb./ton broken
5,000 tons & over 1.25 lb./ton broken

Price - $.20/lb.

NORMAL BUDGET ALLOWANCE

0 - 5,000 ton stopes, 2.00 lb./ton x 2000 tons x $.20/lb. = $ 800
5,000 tons & over, 1.25 lb./ton x 5650 tons x $.20/lb. = $ 1,413
Total Normal Budget Allownace $ 2,213

CURRENT BUDGET ALLOWANCE

$.40 x No. tons produced from 0 - 5,000 ton stopes
+$.25 x No. tons produced from 5,000 tons & over stopes

Figure 2.—Stoping: Operating supplies.

S. & K. MINE

SUMMARY OF STANDARD BUDGET FOR 19

Cost Center Stoping Summary	Standard Unit Tons Ore Produced	Units in Month 7,650

Account Name	Acc't Class	Budgeted Dollars Fixed	Variable	Total	Unit Standard Cost
Op. Labor	V	-	4,766	4,766	.62
Op. Supplies	V	-	2,213	2,213	.29
Other Charges	V	-	2,000	2,000	.26
Total		-	8,979	8,979	1.17

Figure 3.—Stoping summary.

S. & K. MINE

BUDGET DATA FOR 19

Standard Period Month	Code Name Pumping & Drainage	Cost Center General Mine Expense
Date	Standard Unit Tons Ore Produced	Normal Budget $ 2,144

STANDARD DATA

Op. Labor
Usage - 3.0 MS/cal. day
Price - $12.00/MS
Cost - 3 x $12.00 = $36.00/cal. day
Normal month - $36.00/cal. day x 30.4 days = $ 1,094/mo.

Op. Supplies -

Repair Labor
Shops Distribution, 100 hr./mo. @ $2.00/hr. 200/mo.

Repair Supplies
Warehouse Distribution averages 250/mo.

Power
Power Distribution (60,000 KWH @ $.010/KWH) 600/mo.

Other Charges -

NORMAL BUDGET ALLOWANCE $ 2,144/mo.

CURRENT BUDGET ALLOWANCE

Days in Month	Labor $/Cal.day	Labor Dollars	Repair Labor, Supplies & Power	Total Current Budget Allow.
28	36	1,008	1,050	2,058
30	36	1,080	1,050	2,130
31	36	1,116	1,050	2,166

BUDGETARY ADJUSTMENTS

28 day month $ 86
30 " " 14
31 " " -22

Figure 4.—Detail sheet for pumping and drainage.

used in the budget, the allowance for stoping labour will be calculated each month according to the formula shown under *Current Budget Allowance.*

A typical budget for stoping operating supplies is shown in Figure 2. Again, for the sake of simplicity, it has been assumed that the only item of supplies being charged to stoping is blasting powder, and that other items will be included in another class of expense, charged either directly or indirectly as a part of a service supplied. The pattern is similar to that for stoping labour and should require no further explanation.

Other items of cost chargeable to stoping are budgeted in a similar manner. In this example, they are all included under one heading — *Other Charges.* The total dollars appears on the summary of the stoping costs. This summary is made on what is called a budget summary sheet (Figure 3). On this sheet are summarized the normal budget allowances for all classes of expense budgeted in the cost centre. The account class (fixed, variable, mixed, or irregular), and the fixed and variable proportion of the total cost, are shown for each class of expense. If an expense is an 'irregular' type of expense because of seasonal variations, it is shown in the 'fixed' column. If it is 'mixed' because made up of fixed and variable components, each component is shown in its respective column. The normal budget allowances are totaled and the final total is divided by the number of standard units in the normal activity for this cost centre. This is the unit standard cost of the cost centre expressed in terms of the standard unit of activity for that cost centre. In the case of the example, the unit of activity of the stoping cost centre has been taken to be the same as the unit of activity of the mine as a whole. This would not be the case where shrink-

S. & K. MINE

SUMMARY OF STANDARD BUDGET FOR 19

Cost Center: Gen. Mine Expense	Standard Unit: Tons Ore Produced	Units in Month: 7,650

Account Name	Acc't Class	Budgeted Dollars			Unit Standard Cost
		Fixed	Variable	Total	
Air & Water Lines	I	1,500	580	2,080	.27
Ventilation	F	1,770	-	1,770	.23
Telephones, Lighting, Signals	F	405	-	405	.05
Pumping & Drainage	I	2,144	-	2,144	.28
Surface Transportation	F	190	-	190	.03
Total		6,009	580	6,589	.86
Recapitulation					
Op. Labor		1,800	580	2,380	.31
Op. Supplies		410	-	410	.05
Rep. Labor		800	-	800	.10
Rep. Supplies		1,218	-	1,218	.16
Power		1,500	-	1,500	.20
Other Charges		281	-	281	.04
Total		6,009	580	6,589	.86

Figure 5.—Summary sheet, showing normal budget allowances.

S. & K. MINE

SUMMARY OF STANDARD BUDGET FOR 19

Cost Center: Mine Summary	Standard Unit: Tons Ore Produced	Units in Month: 7,650

Account Name	Acc't Class	Budgeted Dollars			Unit Standard Cost
		Fixed	Variable	Total	
Development	V	-	6,100	6,100	.80
Stoping	V	-	8,979	8,979	1.17
Hoisting	I	1,800	545	2,345	.31
Underground Trans.	I	919	4,851	5,770	.75
General Mine Exp.	I	6,009	580	6,589	.86
Total - Direct Mining		8,728	21,055	29,783	3.89
Mine Overhead	F	10,260	-	10,260	1.34
Prop'n. Head Office O/H	F	5,200	-	5,200	.68
TOTAL COST		24,188	21,055	45,243	5.91
Recapitulation of Direct Mining Expense					
Op. Labor		2,020	12,260	14,280	1.87
Op. Supplies		1,061	3,800	4,861	.63
Rep. Labor		920	1,342	2,262	.29
Rep. Supplies		541	653	1,194	.16
Power		3,142	-	3,142	.41
Other Charges		1,044	3,000	4,044	.53
Total - Direct Mining		8,728	21,055	29,783	3.89

Figure 6.—Mine summary sheet.

age stoping methods are used, as the tons of ore broken (which would be the unit of activity for the stoping cost centre) is not necessarily the same as the tons of ore taken from the mine for milling or for shipment. The standard cost for development cost centres would be expressed in terms of dollars per foot of advance.

General Mine Expense Budget

In those cost centres where the budget is to be built up according to codes rather than class of expense, a slightly different arrangement is used in preparing the budget detail sheet for each code. An example of this is shown in Figure 4, where pumping and drainage is one of the codes to be budgeted within the general mine expense cost centre. Under the heading *Standard Data,* each class of expense is shown, together with its normal budget allowance and how it has been determined. Under operating labour it will be noted that the number of days used in the normal month is 30.4, even though the normal number of days worked in the mine is only 25.5. This results from the necessity for a man to be on duty at all times in this work. The *Normal Budget Allowances* for pumping and drainage will be the total of the individual class of expense allowances. Since the labour usage depends on the number of days in the month, there will be a slightly different *Current Budget Allowance* for months of different length, as shown in the Table. These differences are shown in the Table under *Budgetary Adjustments,* which represent the difference between the current budget allowance and the normal budget allowance. It is necessary that these differences be kept separate so that they will not distort the variances on the statement and thus complicate their interpretation.

The normal budget allowances for all codes within the general mine expense cost centre are summarized on a *Budget Summary Sheet* (Figure 5). The unit standard cost for the cost centre is calculated in the same way as for stoping. In addition, the normal allowances are recapitulated according to class of expense. This is done in order that the total cost, for the mine as a whole, of each class of expense can be summarized when all the individual cost centres have been budgeted.

Mine Summary Budget

Figure 6 shows the summary of all the cost centres and is used in arriving at the final budgeted cost of mining a ton of ore. The budgeted dollars on the 'total' line of each cost centre's budget summary sheet is posted to this sheet and the unit standard cost is calculated in terms of tons of ore produced, the unit of activity for the mine as a whole. In this example, the unit cost for stoping and for general mine expense is the same as shown on the mine summary, inasmuch as the unit of activities is the same. In development, however, the standard cost would have been expressed in terms of feet advance whereas on the summary sheet it would be given in terms of tons ore produced. It will be noted that the direct mining expenses have been kept separate from the overhead type of expenses. Usually, this will conform to the lines of responsibility and the recapitulation by class of expense is made only for the direct mining expenses.

Sample Statements

A cost and variance statement for each cost centre, and a mine summary statement, are prepared each month for purposes of measuring actual performance against the budget. In this example, three different statements will be considered — stoping and general mine expense cost centre statements, and the mine summary statement. Discussion will concern primarily the interpretation of the figures and how they can be used by management. The mechanics of preparing the statements and cal-

TABLE II

STOPING: Tons ore produced (tons stoped).....8,400 tons

STOPES: 0 — 5,000 tons.....2,700 tons
5,000 tons and over.....5,700 "

DEVELOPMENT — FEET ADVANCE

ROCK TYPE	DRIFTS	RAISES	TOTAL
A.....	450	—	450
B.....	200	50	250
TOTAL.....	650	50	700

S. & K. MINE

Month	COST AND VARIANCE STATEMENT	For Months to Date
Dept.	Cost Center STOPING	Key Man

Tons	PRODUCTION - Tons Ore Produced	Tons
7,650	Normal Activity	
8,400	Actual Activity	
109.8%	Activity Ratio	
$/Ton	COSTS - $/Ton Ore Produced	$/Ton
1.17	Standard Cost	
1.18	Actual Cost	
- .01	Variance	
	ANALYSIS OF VARIANCE	
-	Budgetary Adjustments	
- .02	Mixture Effect	
.01	Operating Costs	

Budget $	Actual $	Variance $	Variance $/Ton	ANALYSIS OF OPERATING COST VARIANCE	Actual $	Variance $	Variance $/Ton
5,268	5,661	-393	-.05	Operating Labor			
				Contract Labor			
		- 60		Price Variance			
		349		Usage Variance			
		- 99		Contract Earnings			
				Day Pay Labor			
		68		Price Variance			
		-651		Usage Variance			
2,505	2,428	77	.01	Operating Supplies			
		269		Price Variance			
		-192		Usage Variance			
2,184	1,800	384	.05	Other Charges			
9,957	9,889	68	.01	TOTAL			

STATISTICS

	Month Budget	Month Actual	To Date Budget	To Date Actual
Production:				
0 - 5,000 ton stopes	2,000	2,700		
5,000 ton & over stopes	5,650	5,700		
Total	7,650	8,400		
Labor:				
Contract - $/MS	$ 11.00	$ 11.30		
MS	223.5	198.6		
Contract Earnings	$ 3.00	$ 3.50		
Productivity, MS/Ton	.027	.024		
Day Pay - $/MS	$ 11.07	$ 10.80		
MS	193.2	252.0		
Supplies:				
Price - $/lb.	$ .20	$ .18		
Usage - lbs.	12,525	13,486		
lbs./ton	1.19	1.61		

Figure 7.—Cost and variance statement: stoping.

S. & K. MINE

Month	COST AND VARIANCE STATEMENT	For Months to Date
Dept.	Cost Center GENERAL MINE EXPENSE	Key Man

Tons	PRODUCTION - Tons Ore Produced	Tons
7,650	Normal Activity	
8,400	Actual Activity	
109.8%	Activity Ratio - % Actual to Normal	
$/Ton	COSTS - $/Ton Ore Produced	$/Ton
.86	Standard Cost	
.81	Actual Cost	
.05	Variance	
	ANALYSIS OF VARIANCE	
.07	Volume Effect	
-	Budgetary Adjustments	
-.02	Operating Costs	

Budget $	Actual $	Variance $	Variance $/Ton	ANALYSIS OF OPERATING COST VARIANCE	Actual $	Variance $	Variance $/Ton
2,137	2,000	137	.02	Air & Water Lines			
1,770	1,850	- 80	-.01	Ventilation			
405	405	-	-	Telephones, Lighting, Signals			
2,166	2,400	-234	-.03	Pumping & Drainage			
190	186	4	-	Surface Transportation			
6,668	6,841	-173	-.02	TOTAL			

ANALYSIS OF MONTH'S COSTS BY CODES

Code Name	Operating Lab.	Operating Supp.	Repair Lab.	Repair Supp.	Power	Other Charges	Total $	Total $/Ton	To Date $/Ton
Air & Water Lines	800	380	228	552	-	40	2,000	.24	
Ventilation	104	214	180	392	900	60	1,850	.22	
Telephone, Lighting & Signals	290	35	-	-	-	80	405	.05	
Pumping & Drainage	1240	-	300	260	600	-	2,400	.28	
Surface Trans.	66	-	-	-	-	120	186	.02	
TOTAL $	2500	629	708	1204	1500	300	6,841		
$/Ton	.30	.07	.08	.14	.18	.04		.81	

Figure 8.—Cost and variance statement: general mine expense.

culating the variances will not be discussed except as it seems desirable for a better understanding of the statement. The preparation of the statements and their supporting work sheets is usually described in detail in a set of standard practice instructions supplied to the accounting department.

Actual Activities

The actual activities assumed for the month are as given in Table II.

Cost and Variance Statement Stoping

A cost and variance statement for stoping is shown in Figure 7. Under *Production,* the actual number of tons produced during the month is seen to be 109.8 per cent of the normal activity. The unit costs in the next section, however, indicate that the actual cost has exceeded the standard cost by $0.01 per ton of ore produced. (Note: Negative or unfavourable variances are normally shown on the cost statements in red, but for purposes of reproduction they are shown in these examples with a minus sign before them). A breakdown of this $–0.01 variance between budgetary adjustments, mixture effect, and operating costs is shown under *Analysis of Variance.* There were no budgetary adjustments to be made in this cost centre.

The mixture effect variance is seen to be unfavourable to the extent of $0.02 per ton. In the production section of the statistics shown at the bottom of the statement it is seen that a greater than normal proportion of the work done during this particular month has been in the 0-5,000-ton stopes. These are the high-cost stopes in that the budgeted productivity in terms of tons produced per man-shift is less than in the 5,000 tons and over group. In addition, the budgeted usage of powder is greater in the smaller stopes. The mixture effect variance represents the effect on costs of the elements of actual work done differing, in proportion to the total, from the forecast proportions of work to be done. An unfavourable mixture effect variance in any month is not necessarily a bad condition. If a trend is indicated by the 'to-date' costs (on the right-hand side of the statement), then a further study should be made, if necessary, to determine why the programme of work for the year is getting so far out of line of the budgeted forecasts.

The operating costs variance is seen to be favourable to the extent of $0.01 per ton. The next section of the statement shows an analysis of this variance. The costs and variances are arranged in the order in which the budget was prepared. The figures in the *Budget* $ column are the current budget allowances as determined from the budget detail sheets in the budget for each class of expense. The actual cost is shown against the budgeted cost and the variance is calculated as the *Budget* $ less the *Actual* $. The variance is shown in terms of dollars and dollars per ton. It is seen immediately that even though the overall operating cost variance for the month was $0.01 per ton favourable, the operating labour variance was $0.05 per ton unfavourable. This was off-set by a favourable variance of like amount in the 'other charges' class of expense. The operating labour variance should be examined further.

The operating labour variance is made up of five parts: price, usage, and contract earnings variances on

contract labour, and price and usage variances against the day-pay labour.

A price variance is that part of the total variance which is caused by the price or dollars per man-shift being different from the figure used in the budget as standard. It may result in the case of labour from the basic rates per hour paid the men having changed since the budget was prepared or from the mixture of high and low priced classes of labour being in different proportion from that used in the budget. In the case of materials or supplies, a price variance results when the actual dollars paid per unit of material or supply differs from the standard price.

A *usage variance* is that part of the total variance resulting from a different number of man-shifts or units of supplies being used than is standard for the volume of work done. In most cases, usage variances are considered controllable and, if the variance gets too far out of line, either favourably or unfavourably, an explanation should be required of the key man responsible for the costs of the cost centre in question.

In the case of contract labour there is an additional variance resulting from the contract miners earning more or less than a recognized average earning per man-shift over and above the basic guaranteed day's pay. Thus, if the stope miners have earned more than the budgeted average earnings, the variance for contract earnings will be unfavourable in that more dollars have been spent than was budgeted. This is a favourable condition, however, in that the higher earnings were attained as a result of more tons than standard produced per man-shift. This, then, would be reflected by a favourable usage variance for contract labour. Reference to the data at the bottom of the statement bears out this reasoning. It is seen that contract earnings were $0.50 per man-shift greater than budgeted and that the number of man-shifts used was only 198.6 instead of the allowed 223.5 for the 8,400 tons stoped during the month. Or, in other terms, the man-shifts used per ton produced were 0.024 as against the 0.027 budgeted.

The usage variance of the day-pay labour is quite unfavourable and the statistics indicate that 252.0 man-shifts were actually used whereas only 193.2 were allowed. An explanation of this difference would certainly be in order.

The total variance for operating supplies (blasting powder) is favourable. Upon further examination, however, it is apparent that this was caused by a favourable price variance, the result of the company being able to purchase the powder at a lower price than was budgeted. The savings realized from this source have been counteracted to a large extent by the unfavourable usage variance. A study of this brought to light the fact that, in the smaller stopes (0—5,000 tons), 2.25 lb. of powder was used per ton of ore produced instead of the standard allowance of 2.00 lb. per ton. This may have been part of the reason for the contract miners showing a more favourable productivity during this period. In any case, by reference to the budget, to the variances shown on the statement, and to the associated statistics, management is provided with the necessary facts and information to enable them to make decisions with confidence.

Cost and Variance Statement General Mine Expense

A cost and variance statement for the general mine expense cost centre is shown in Figure 8. The same general format is used as for stoping but with a different type breakdown of the operating cost variance and with the addition of the costs which go to make up the total actual cost of each code shown by class of expense.

The analysis of the total variance ($0.05 per ton) includes no mixture variance but introduces a very important variance termed *volume effect*. The volume effect arises from spreading the fixed costs over the number of units of current activity rather than over the normal activity. Since all budgeted costs in the stoping cost centre were variable in nature, there was no effect on unit cost resulting from changes in activity. In the general mine expense cost centre, however, the largest part of the costs have been budgeted as fixed, and the fluctuations in activity from month to month are an important factor in the analysis of the unit costs. This will be illustrated graphically a little later.

In the analysis of the operating cost variance section of the statement, a glance at the variance column indicates that the pumping and drainage costs for the month are high and that a further investigation of them is desirable. The total actual cost of $2,400 is detailed further by class of expense in the bottom section of the statement. Comparison of each of these charges against the allowance as shown on the budget detail sheet for this code and for a 31-day month tells us that operating labour was $124 over, repair labour was $100 over, and reamount allowed for this month. The key man for this cost centre had decided during the month to do some repairs that were desirable from a long-range standpoint but had not been foreseen when the budget was prepared. This explanation was considered adequate by management. The operating labour variance, however, he could not explain at first glance or recollection. With very little extra effort, he could determine how much of this variance was due to extra man-shifts worked and how much to higher rate per man-shift paid for overtime or other reasons. This sort of analysis would be made only for those items of cost which are very far out of line.

Variance Relationships

The relationship of the standard cost, actual cost, total variance, volume effect, and operating costs variances is illustrated graphically in Figure 9. The scale of the graph has been distorted purposely to illustrate better the different relationships.

The vertical scale represents dollars and the horizontal scale tons of ore produced. Any point between them will be in terms of dollars per ton of ore produced. The standard cost of $0.86 per ton is arrived at in the budget by adding the fixed costs ($6,009 at all activities) and the variable costs at the normal activity.

The current budget allowance at any activity is represented by a line connecting the fixed costs at zero activity with the total costs at normal activity. The actual activity is represented by the dashed line. The current budget allowance dollars at actual activity divided by the actual activity of 8,400 tons gives a unit cost of $0.79 per ton, or $0.07 per ton lower than the standard cost. A line has also been drawn on the chart representing the standard cost times the tons. This is a line connecting the origin with the standard cost at the normal activity. The actual cost for the month is $0.81 per ton, which is a point between the current budget allowance and the standard cost on the actual activity line. The variance relationships can then be summarized as:

VOLUME EFFECT
= Stand. cost − C. B. A.
= 0.86 − 0.79 = +0.07

OP. COST VAR.
= C. B. A. − Actual cost
= 0.79 − 0.81 = −0.02

TOTAL VARIANCE
= Stand. − Actual cost
= 0.86 − 0.81 = +0.05

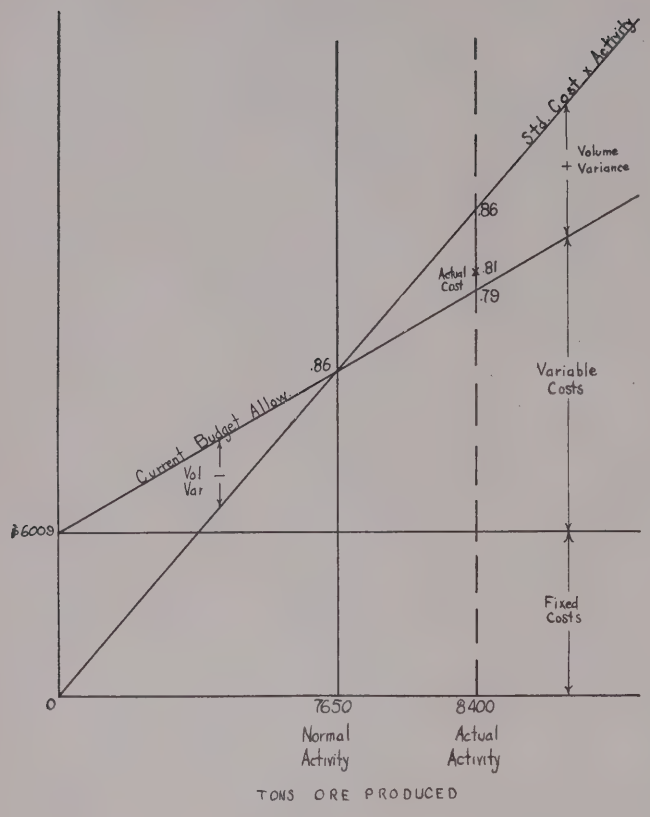


Figure 9.—Graph showing relationship of standard cost, actual cost, total variance, volume effect, and operating costs variances.

S. & K. MINE

Month January 1952	Summary Total Mine Costs	To Date

Tons	PRODUCTION - Tons Ore Produced	Tons
7,650	Normal Activity	
7,800	Forecast Activity	
8,400	Actual Activity	
109.8%	Activity Ratio - % Actual to Normal	
$/Ton	COSTS - $/Ton Ore Produced	$/Ton
5.91	Standard Cost	
5.65	Actual Cost	
.26	Variance	
	ANALYSIS OF TOTAL VARIANCE	
.28	Volume Effect	
-.02	Budgetary Adjustment	
.09	Mixture	
-.09	Operating Costs	

ANALYSIS BY COST CENTERS

Month Variances Total	Oper. Cost	Mix.	Bud. Adj.	Vol. Eff.	Act. Cost	Std. Cost	Cost Centers $/Ton	To Date Std. Cost	Act. Cost	Variances Vol Eff	Bud Adj	Mix	Op Cost	Tot
.11	.03	.11			.66	.80	Development							
-.01	.01	-.02			1.18	1.17	Stoping							
.05	.04		-.01	.02	.26	.31	Hoisting							
-.06	-.06		-.01	.01	.81	.75	Underground Transportation							
.05	-.02			.07	.81	.86	General Mine Expense							
.17		.09	-.02	.10	3.72	3.89	Total-Direct Mining							
.03	-.09			.12	1.31	1.34	Mine Overhead							
.06				.06	.62	.68	Prop'n-Head Office Overhead							
.26	-.09	.09	-.02	.28	5.65	5.91	TOTAL							

Figure 10.—Summary, total mine costs.

It should also be noted that the volume variance is always favourable at activities above the normal activity and unfavourable at activities below the normal activity.

Mine Summary Statement

The costs and variances for all the cost centres in the mine are summarized on the mine summary statement (Figure 10). It will be noted in the production section at the top of the statement that a forecast activity has been shown. The normal activity is based on 300 tons a day for 25.5 days. The forecast activity is 300 tons a day times the actual number of days to be worked during the month, in this case 26. The difference between the normal activity and the forecast activity is due strictly to the number of days in the month and not to performance. The difference between the forecast activity and the actual activity is a measure of how well the production forecast for the month has been met.

The figures shown in the *Costs* and the *Analysis of Total Variance* sections are total figures for the mine as a whole. They are shown broken down by cost centres in the *Analysis by Cost Centres* section of the statement. On this statement, all unit costs are expressed in terms of tons of ore produced even though the unit of activity of a particular cost centre may be something different. In this example, development is the only cost centre where the unit of activity is different, its costs being expressed in terms of feet advance. In addition to the mixture variance which arises within the development cost centre, as described earlier, another mixture variance can occur where the development costs are summarized with the other costs of the mine. This will occur when the proportion of development costs to the total mining costs differs from the proportion to the total used in the normal budget. If, for instance, the development work in the mine began to lag appreciably from the programme established at the first of the year but other costs were at their normal levels, then a favourable mixture variance would appear on the mine summary statement, simply because dollars had not been spent that it had been planned should be spent. If this condition were allowed to continue for any length of time, serious consequences might result, particularly where actual mining of the ore follows closely on the heels of development work.

Coal Preparation Progress in Western Canada

*By H. WILTON-CLARK**

(*Annual Western Meeting, Calgary, Alta., October,* 1951)

(*Transactions, Volume LV,* 1952, pp. 69-73)

UP TO thirty years ago, coal preparation in western Canada, except at four plants, had not included actual coal washing. The exceptions were at Nanaimo and Royston, B.C., where tub washers and wet tables were in use; at Hosmer, B.C., using Robinson cones; and at Leitch Collieries, Passburg, Alta., using a feldspar jig. The two last named collieries had a short life and were not operating in 1920.

The industry was divided, as at the present day, into two sections, steam (bituminous) coal and domestic (largely sub-bituminous) coal, the first generally supplying industrial needs and the second household and other heating requirements. The two sections overlap to some small degree today as prepared sizes, chiefly stoker, and briquettes, from steam coal are shipped for household use, and domestic slacks are used in certain industrial plants.

The domestic operator, for reasons shown later, was not primarily concerned with actual coal washing as a phase of coal preparation, and bent his efforts to improvement of screening and loading plants, with hand-picking of coarse sizes. The bituminous operator, on the other hand, felt it necessary by 1920 to consider coal washing.

When the mines were comparatively young, coal quality could be controlled to a considerable degree by selective mining, by leaving behind some of the seam or by-passing districts in a mine where the coal was of inferior quality to that in other districts. This obviously had its limitations, and so the next step was coal washing. Merely listing plants by name, and mentioning types of washing equipment incorporated in them, is of no particular interest or value, but the general growth and progress of coal washing does present an interesting pattern in the light of the problems presented to the operator.

The steam coal mines produce coal ranging from low to high volatile bituminous. The mines are all located in the mountains, chiefly in railway-traversed passes. The coal is all found in the Kootenay and Luscar formations, and has attained its rank due to the tremendous pressures exerted by mountain building forces. In almost all occurrences, the seams are pitching, ranging from 15 to 90 degrees, and they are often folded, faulted, and to some degree irregular. The coal has no marked cleats and it lacks structure. Usually about half the coal, as mined, will pass through a ¼-inch square screen. Figure 1 shows a sizing curve for such a coal.

This has an important bearing from a washing viewpoint because of the greater difficulty, and consequent higher cost, of washing fine slack compared to the cost of washing coarser sizes. This cost comparison is even more unfavourable if the slack is wet-washed, and then, of necessity, dried. It has often been stated that a given piece of coal washing equipment does not wash tons of coal per hour but a certain number of particles per hour, and this is almost true. The same remark might be made about drying equipment, in view of the much higher quantity of superficial moisture carried by fine coal than by coarse. Size characteristics, however, are not alone responsible for the difficulty encountered in washing western steam coals. Specific gravity distribution of the coal makes for an exceptionally difficult washing condition. Figure 2 shows cumulative float-ash curves for four coals. *A* and *B* are bituminous coals from Pennsylvania, chosen because they are easy to wash, and *C* and *D* are two Alberta bituminous coals, chosen because they are probably the two Western Canadian coals most difficult to wash. Assuming that each coal is to be washed at 1.6 specific gravity, the theoretical ash content and percentage recovery of these coal would be:

Coal	% Ash	% Recovery
A.....	8.05	94.7
B.....	5.12	93.0
C.....	10.69	87.2
D.....	11.30	81.5

Now consider Figure 3, which contains the same data plotted to show specific gravity distribution.

The curves are obtained by plotting the weight percentage of material within the range of 0.10 specific gravity on either side of a chosen specific gravity point, against the chosen specific gravity. Thus the weight percentage of material lying in the specific gravity range 1.30 to 1.50 is plotted against specific gravity 1.40, and so on.

Figure 4 shows a second set of specific gravity distribution curves for the same coals, but using the range of 0.15 specific gravity.

The reason for showing the curves is that, years ago, a prominent American coal preparation engineer, B. M. Bird, came to the conclusion that the degree of difficulty of wet-washing coal is largely determined by the specific gravity distribution, as indicated in Table I.

On this basis, the wet-washing characteristics of the coals *A*, *B*, *C*, and *D* at specific gravity 1.6, determined from the curves shown in Figure 3, would be as indicated in Table II.

As previously mentioned, the washing difficulty is further increased by the high percentage of fines in the Alberta coals and the high cost of subsequent drying. The total cost is such that, to date, the mines have been forced to limit wet washing and drying to plus ¼-inch sizes.

The fines are usually cleaned by air equipment, consisting of air tables or air jigs. Bird's conclusions

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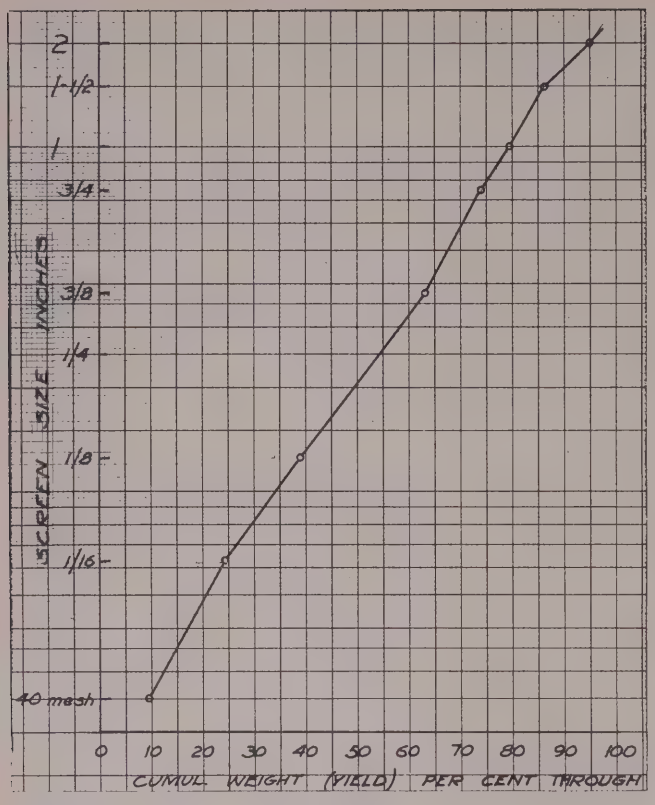

Figure 1.—Typical screen analysis of Alberta bituminous (steam) coal.

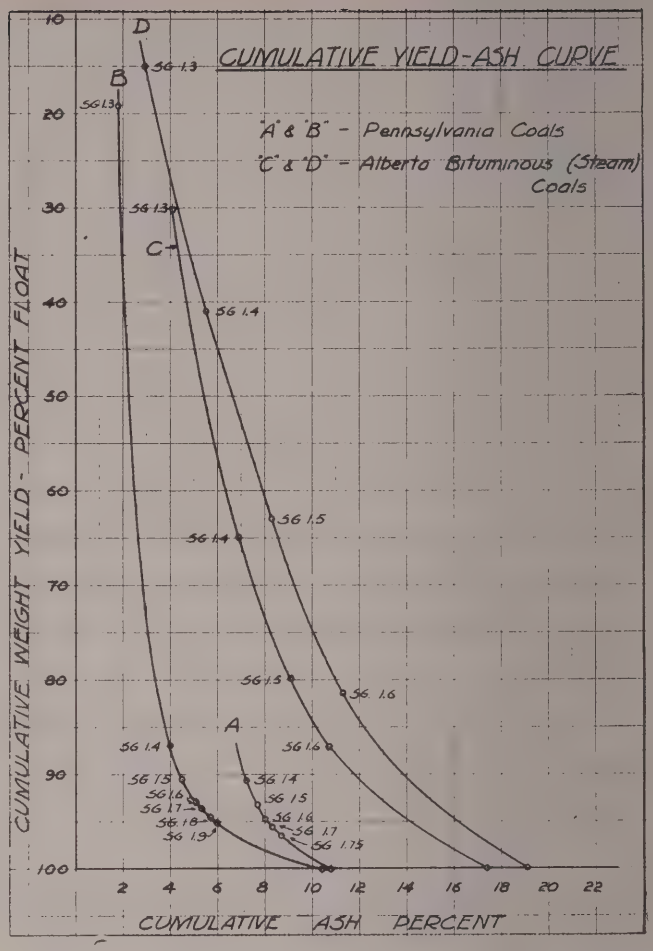

Figure 2.

as to the degree of difficulty of wet washing can be applied to air cleaning within certain limits but, instead of the ±0.10 specific gravity distribution curves, those for ±0.15 specific gravity distribution (Figure 4) should be used. Table III gives the data so obtained for the air cleaning at specific gravity 1.6 of these Pennsylvania and Alberta coals.

Unfortunately, due to the low density of the washing medium, no air cleaners can be classed as 'exceptionally efficient', and it follows that, in practice, only rock or very heavy bone can be removed from the coal by such methods, unless the operator is prepared to accept very low washing efficiency, with resultant high refuse losses.

To obtain best possible results within the economic possibilities, operators have been raising efficiencies of air cleaning methods by close sizing of feeds to air cleaners, and by a further step, now being investigated by some and already in use by others, of pre-drying the raw coal. The efficiency of air cleaners, when working on fine coal, is greatly affected by small changes in superficial moisture content of the coal, and dry feed insures highest possible efficiency and at the same time eliminates freezing trouble with the coal as shipped. It may appear that the acceptance of pre-drying of raw coal is at variance with earlier

TABLE I.— RELATIONSHIP OF SPECIFIC GRAVITY DISTRIBUTION TO EASE OF WET WASHING
(*After B. M. Bird*)

±0.10 Sp.G. Curve	Degree of Difficulty of Wet Washing	Preparation
2—7 per cent. . .	Simple	Almost any process; high tonnage
7—10 " " . . .	Moderately difficult	Efficient process; high tonnage
10—15 " " . . .	Difficult	Efficient process; medium tonnage
15—20 " " . . .	Very difficult	Efficient process
20—25 " " . . .	Exceedingly difficult	Very efficient process; low tonnage
Above 25 " " . . .	Formidable	Limited to a few exceptionally efficient processes

TABLE II.— WET WASHING CHARACTERISTICS OF COALS A, B, C, and D AT SP. G 1.6

±0.10 Sp.G. Curve (Figure 3)		Degree of Difficulty of Wet Washing	Preparation
Coal	Per Cent		
A.	2	Simple	Almost any process, at high tonnage
B.	3	Simple	Almost any process, at high tonnage
C.	11.5	Difficult	Efficient process, at medium tonnage
D.	28	Formidable	Limited to a few exceptionally efficient processes

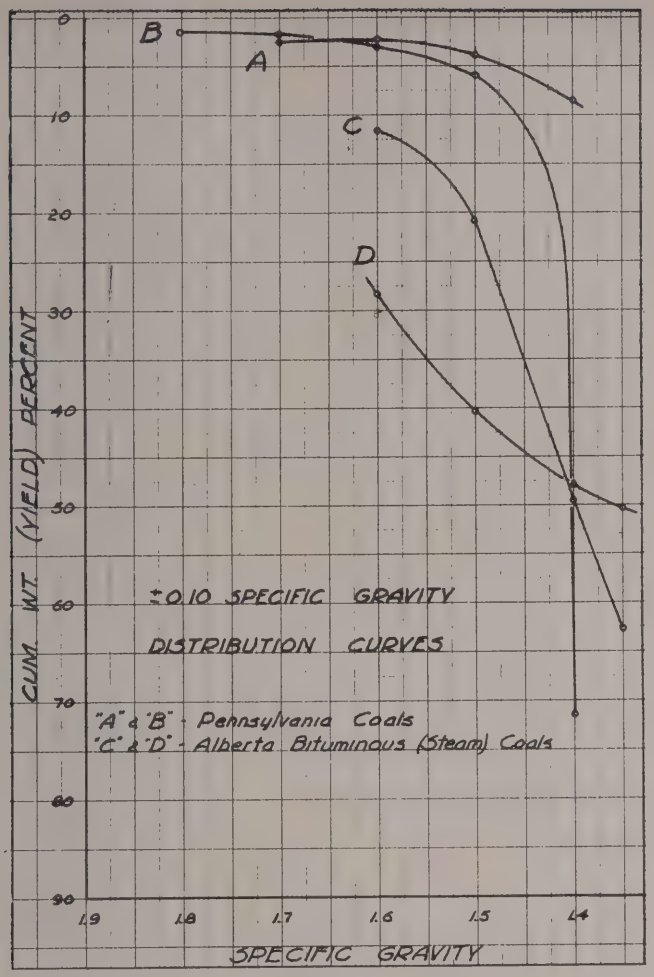

Figure 3.

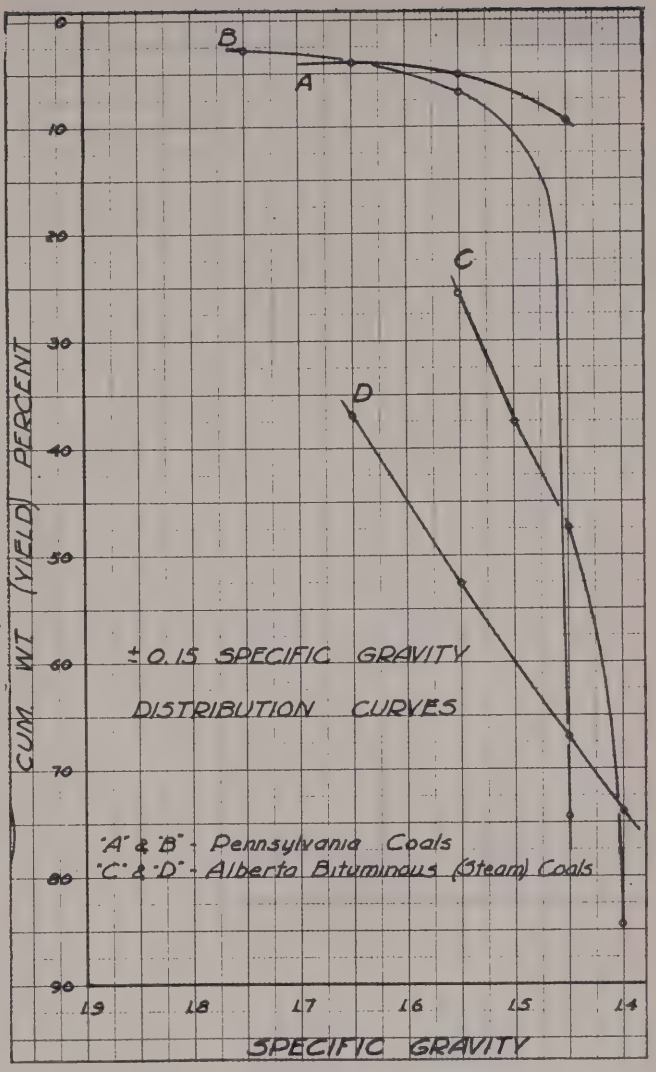

Figure 4.

remarks about the high cost of drying wet-washed coals, but this is a matter of degree. Pre-drying fine coal to the extent of removing 2 or perhaps 3 per cent of its moisture is a different matter from de-watering the fines after wet washing and then drying from 18 to 20 per cent moisture content to a final 1 or 2 per cent.

No matter whether wet washing or air cleaning is used, no strictly mechanical process has yet cleaned coal of the size range −30 mesh. As far as is known today, the only effective method of treating such coal is to follow metal mining concentrator practice of froth flotation with subsequent filtering to produce a cake, followed by heat drying. From a qualitative point of view, such cleaning of coal dust can be effectively carried out, but it is doubtful if costs will admit of such practice in the next few years. There is a modification to the foregoing statement in that such treatment may be reasonable for the limited quantity of fines produced due to degradation when washing coarser sizes.

Western domestic coals range from the lignites of Saskatchewan through sub-bituminous coals such as mined at Drumheller, to bituminous coals as at Lethbridge and Coalspur. As the rank of coal becomes higher, inherent moisture decreases and calorific value increases. All the coals are hard and blocky as mined and exhibit varying degrees of slacking with exposure to air, from a low with the lignites to virtually no slacking over long periods with Coalspur coals.

Excellent physical and chemical studies of our western coals have been made in the Fuel Research Laboratories of the Bureau of Mines, Ottawa, and the results of these have been made available to the operators. The following data and remarks on a Drumheller coal are quoted or summarized from a report by Mr. E. Swartzman of the Bureau of Mines.

Figure 5 gives the screen analysis of the coal as mined.

Figure 6 shows cumulative yield-ash curves for 4 in. lump, 4 in. by 1½ in. egg and nut, and minus 1½ in. stoker and slack.

TABLE III.— AIR CLEANING CHARACTERISTICS OF COALS A, B, C, and D AT SP.G 1.6

±0.15 Sp.G. Curve (Figure 4)		Degree of Difficulty of Air Cleaning	Preparation
Coal	Per Cent		
A.....	4	Simple	Almost any process, at high tonnage
B.....	5	Simple	Almost any process, at high tonnage
C.....	20—25(?)	Exceedingly difficult	Very efficient process, at low tonnage
D.....	Over 25	Formidable	A few extremely efficient processes

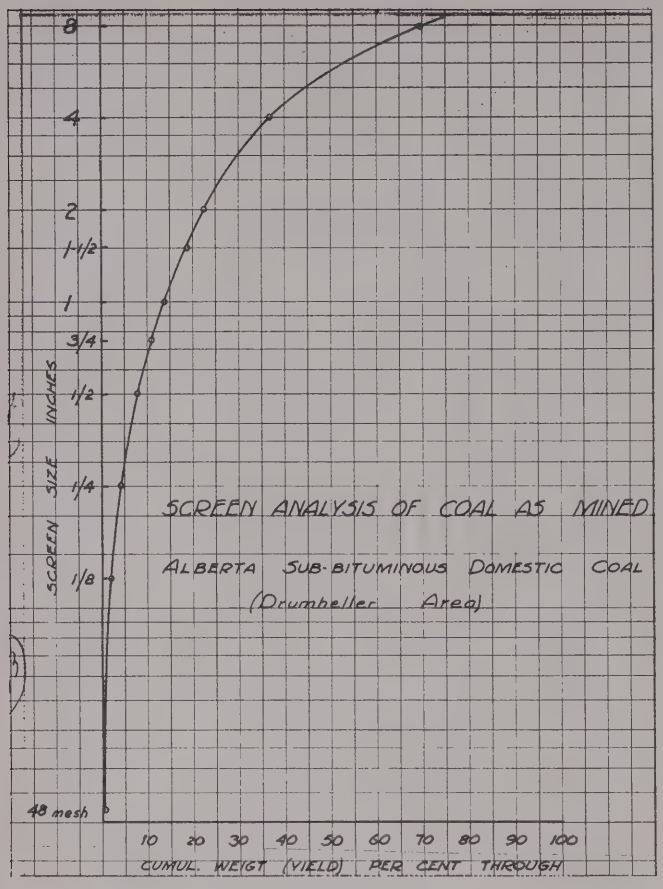

Figure 5.

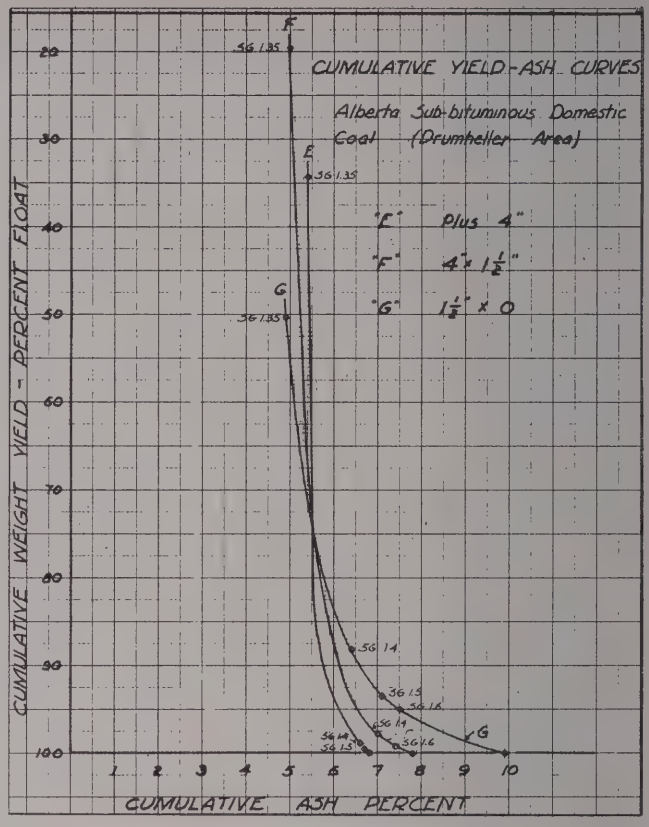
CUMULATIVE YIELD-ASH CURVES
Alberta Sub-bituminous Domestic Coal (Drumheller Area)
"E" Plus 4"
"F" 4" x 1½"
"G" 1½" x 0
CUMULATIVE WEIGHT YIELD - PERCENT FLOAT
CUMULATIVE ASH PERCENT
1 2 3 4 5 6 7 8 9 10

Figure 6.

Figures 7 and 8 are specific gravity distribution curves for the coal on a ± 0.10 and ± 0.15 basis, respectively.

If, again, 1.6 be chosen as the specific gravity of separation, Figure 6 shows that so little improvement could be made in coal coarser than 1½-in by any washing process that it is not worth considering, and that any heavy rock particles could be removed by hand picking; this is why I mentioned earlier that domestic operators have not been primarily interested in coal washing.

The above statement should be modified in that, at times, it is inevitable that roof rock or floor material comes out with the coal, especially with mechanized mining, so that some simple form of washing equipment might be of use on the size range where hand picking becomes difficult, say 3 in. by 1½ in. Such washing equipment would require to be a wet washer, as no air equipment known today is effective on coal larger than 1½-inch.

In the case of minus 1½-inch slack, washing at specific gravity 1.6 would make an effective ash reduction with comparatively small refuse loss, *i.e.*, ash reduction from 9.9 per cent to 7.5 per cent ash with a theoretical refuse loss of 4.9 per cent. The important point for the operator is not that his market requires this ash reduction but that it does or may require the removal of free impurities. Further, considering both the ± 0.10 and ± 0.15 specific gravity distribution curves, Bird's scale would show either a simple wet washing or air cleaning plant to be effective although, on the coarser end of the size scale, loss of coal with the refuse is almost inevitable if air cleaning is used.

In 1920, it is probable that few operators in western Canada had much knowledge of coal washing. Wet washers were being used extensively in Europe and to some extent in the United States, but general information was lacking. Wet washing obviously entailed drying, and dryers were confined to the rotary kiln type which were costly, had low capacity, and enhanced breakage of friable coals. In the United States about this time, air tables which had been developed for seed cleaning were applied to coal cleaning. The air table seemed to be the possible answer to cleaning bituminous coal, especially in our climate. The inherent defects of an air medium as against water or heavy media equipment were not too well realized, and so preparation plants were built in the Crow's Nest Pass area in the 'twenties' until nearly all mines were provided with them. Brazeau built an air cleaner at this time, and, later, mines on the Alberta Coal Branch installed air cleaning equipment.

Much work on coal preparation was being carried out in other coal producing countries until, today, a plant can be designed to produce virtually any desired results within the washability limits of a given coal and compatible with overall economics of production. Such was not the case at the time of the first plants; progress was by the expensive route of trial and error. It became reasonable, then, for plants to develop as they did — first air cleaners; then an attempt at improvement by more cleaners, with each machine working on a small size-range; then introduction of wet washers, first simple rising-current washers, and later jigs of a more complicated type, but far more efficient, until most plants had either Vissac single-cell jigs or four- or five-cell Baum-type washers. At present, at least one mine has displaced jigs in favour of a heavy media process, and it is likely that, for coals with difficult washing characteristics, heavy media equipment will be installed in more and more plants, although some coals will continue to be efficiently washed with jigs. As wet washing came in, the coarser sizes were wet-washed while air cleaning continued to be

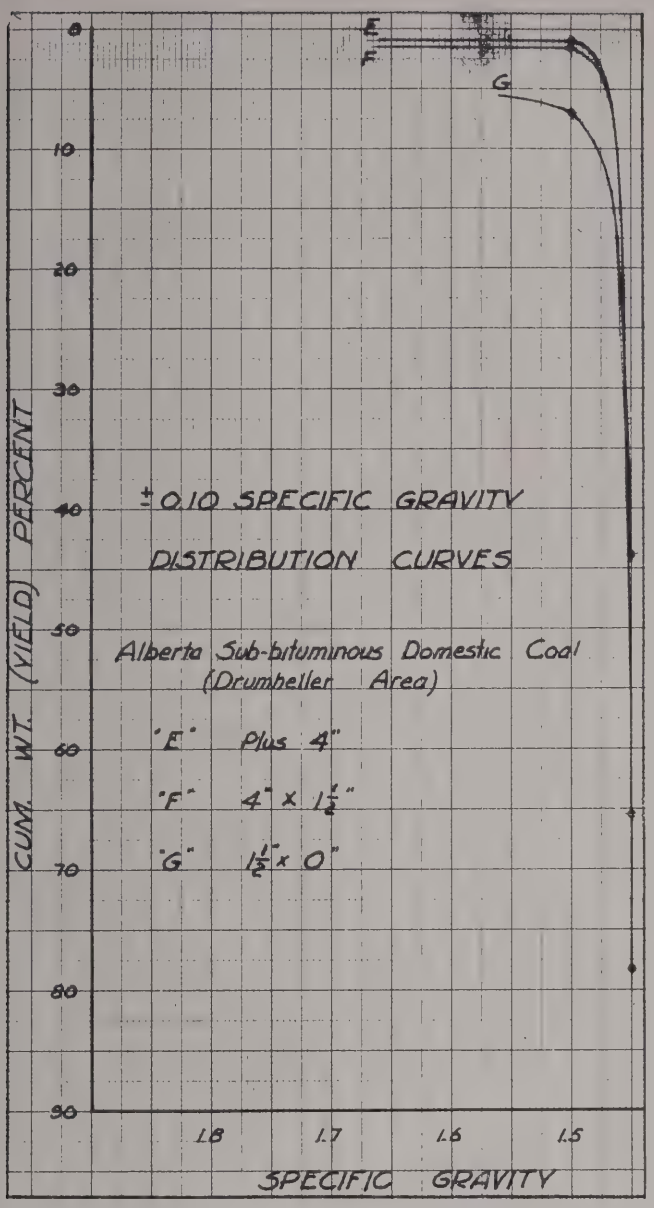

Figure 7.

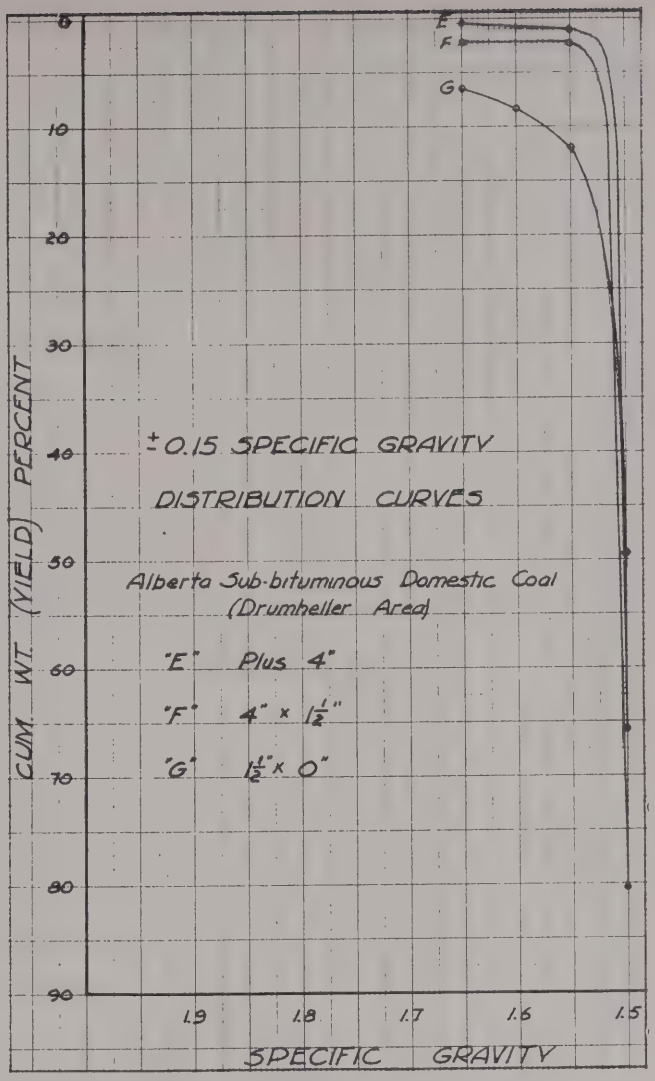

Figure 8.

used on fines. The reason was the drying difficulty and the increase in the cost of drying as the size of coal to be dried decreased. Nevertheless, dryers were improved and increasing tonnages of coal were wet-washed until, by about 1940, it became economically feasible to wet-wash coals down to about ¼-inch size, or about one-half the total output of the average bituminous mine. Dryers, either of the up-draft shaker type or of the 'flash' type preceded by centrifuging, have been developed in the past few years which may allow even smaller sizes of coal than ¼-inch to be wet-washed economically; and if so, this will undoubtedly be the trend. Where air cleaning of fines must still be carried on, pre-drying will become of greater importance, as evidenced by installations already made.

For the domestic coals, it would appear that washing will be confined to nut, stoker, and slack sizes, with wet washing, air cleaning, or both, dependent entirely on the requirements of the individual mine and the economic advantages to be gained.

Because of rising costs, washing on certain sizes may in some instances result in actual monetary loss, but it will be necessary to take this loss in order to retain markets which are becoming increasingly competitive and in which other sizes can be sold at a profit.

Plant installations and improvements will continue, as is right and proper, but the qualitative aspect of such plants must definitely fit in with the economic. However, when the overall difficulties of preparation are considered, it must be allowed that the industry, generally, has accomplished much. Through technical literature and engineering investigation, most companies keep abreast of advances in coal preparation, and will not be slow in taking advantage of methods which can be adapted to their particular problems.

Present Position and Prospects of the Coal Burning Gas Turbine

*By DONALD L. MORDELL**

(*Annual Western Meeting, Calgary, Alta., October,* 1951)
(*Transactions, Volume LV,* 1952, pp. 74-78)

Introduction

A VERY LARGE PART of the world's fuel resources are in solid form, as various grades of coal, but an increasing proportion of fuel demands on this Continent and elsewhere are having to be met by liquid fuels. Convenience and, in many cases, cost weigh heavily in favour of the liquid fuels, and there are many fuel burning applications where the use of coal would be impractical, such as for automotive or air transport. By extrapolating the present trends one might conclude that, in future years in Canada and the United States, coal will be used only in a few power stations close to coal fields. If such a state of affairs should arise, it will obviously prove embarrassing to those accustomed to make their livelihood from the winning and marketing of coal, but it is perhaps not so obvious that it must also prove an embarrassment to the nation.

The major reasons for this are two in number. Firstly, if present coal production is to be replaced by oil production, a large capital expenditure is necessary, which must inevitably be paid by the consumer; and secondly, in times of emergency, large amounts of oil fuels are required for military purposes which cannot be achieved with solid fuels. It is obviously unwise, therefore, to discontinue the use of coal for those non-military jobs that it can do.

If we examine the major fuel markets we find that there are three:

Domestic heating
Steam generation
Transport by air, sea, and land

For domestic heating, the extreme convenience of oil, in spite of its greater cost, has captured much of the coal market. For steam generation, the choice between oil and coal rests largely with local conditions as reflected in fuel cost; and oil is largely used at sea. For air and road transport, oil reigns supreme, but it is particularly ironic that the coal so long used for rail transport is, in the United States and to a lesser extent in Canada, being replaced by oil burned in a Diesel engine. This engine was invented in the first place specifically for burning coal, although the solid-fuel reciprocating engine has not yet reached, and probably never will reach except in very small sizes, commercial application.

Until a decade ago, the only method of converting the energy latent in coal into mechanical work (other than the commercially insignificant powdered-fuel Diesel engine and the Stirling hot-air engine) was to burn it and produce steam or mercury vapour which can drive an engine. In the last ten years, however, a new type of power plant, the gas turbine, has come into commercial prominence. Its development accelerated greatly by its usefulness as a military power plant, the gas turbine was developed primarily to burn oil. However, unlike the Diesel or gasoline engine, the gas turbine offers a reasonable prospect of being able to utilise coal and so may have an important part to play in the utilisation of solid fuel resources. This paper is concerned with the development of such a coal burning gas turbine.

The Gas Turbine

The gas turbine, natural child of the steam turbine and the internal combustion engine, was first seriously studied and experimented with some fifty years ago. The early experiments were not successful, due primarily to inadequate aerodynamic and metallurgical knowledge. The goal toward which the early inventors were striving was the combination of the smoothness of the steam turbine with the convenience and simplicity of internal combustion.

The commercial development of the gas turbine has now been proceeding for some twenty years. For the first ten it was largely in the hands of one concern, the Brown Boveri Company, of Switzerland, and to their credit stands a long list of 'firsts'. The war, however, stimulated the development and made available funds on a very large scale. Our knowledge of the basic sciences and applied technology was increased to such an extent that, immediately after the war, a large number of organizations embarked on the development of commercial, non-military gas turbines, the first of which are now putting up very good records in service.

What, then, is the gas turbine? Fundamentally, it is no different from any other type of heat engine. A working fuid is compressed, heated, and then expanded. The work obtained in expansion, if greater than that absorbed in the compression process, supplies the useful output which may be harnessed to a generator or propellor or some other load. The working fluid, after expansion, may either be rejected, so that the compressor is continually inspiring fresh working fluid, permitting the use of internal combustion, or it may be cooled and re-admitted to the compressor, in which case it must be heated externally. Now this description of events is applicable to all known heat engines. It is only when we add the stipulations that the working fluid is at all times in the gaseous state and that its compression and expansion are performed in rotating, as distinct from reciprocating, machinery, that we define a gas turbine.

Figure 1 shows diagrammatically the simplest possible gas turbine. The compressor *A* inspires atmospheric air and compresses it to somewhere between three and six atmospheres. It is heated by internal combustion in the chamber *B*, where fuel is injected and burned. The hot

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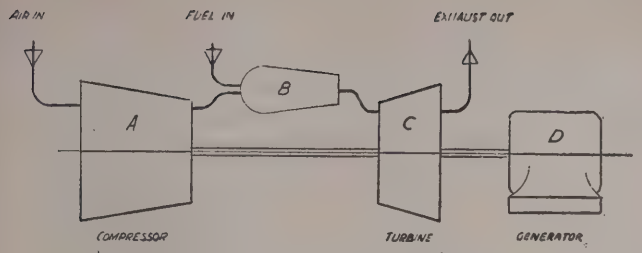

Figure 1.—Simple gas turbine.

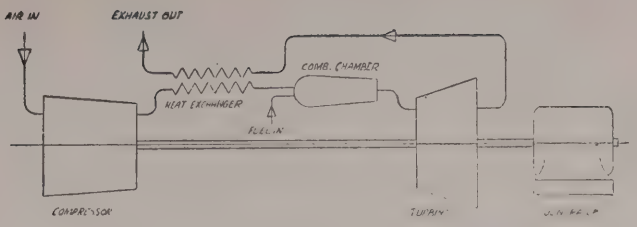

Figure 2.—Gas turbine with heat exchanger.

products of combustion expand in turbine *C*, and, if the design is correct, produce enough power in so doing to drive both the compressor and the load *D*. Such is the simplest gas turbine. Its magic is that there is no large boiler or condenser and, since the working fluid is handled continually by turbo-machinery, very large amounts of air can be handled by compact machinery, resulting in a very light and compact power plant.

There are a great many different ways in which a gas turbine can be arranged, but at this point we should like only to refer to that shown diagrammatically in Figure 2, where it will be noticed that a heat exchanger is employed to transfer some of the heat, that would otherwise be discharged up the stack, to the air leaving the compressor. The use of the exchanger results in a marked gain in fuel economy, at a certain cost in bulk and complexity.

Some idea of the present position of the gas turbine as a prime mover can be presented briefly. In Switzerland, units of up to 27,000 k.w., with 34 per cent thermal efficiency, are running. In the same country, there is ten years of fairly successful operation of a gas turbine locomotive. In the United Kingdom, several large plants are just about to run, and at the time of writing the first large marine gas turbine is undergoing sea trials in a tanker. In the United States, the successful operation of a gas turbine locomotive prompted the railroad to order ten additional units, and some very successful experience has been obtained in a power station in Oklahoma.

In all the applications, the gas turbine can be used to full advantage in cold weather. It is characteristic of the gas turbine that its output and thermal efficiency markedly improve as the temperature drops. This suits many natural load conditions.

The Problem of Using Coal and Its Solutions

In the gas turbine engine, fuel is burned in order to raise the temperature of the air leaving the compressor to the highest temperature that can be tolerated by the materials used in the turbine. In most oil or gas turbines so far used, this heating is accomplished by direct internal combustion. Some of the oxygen of the air is used to burn the fuel. Since the properties of the turbine materials restrict the maximum temperature there is usually a large excess of air. In practice, the fuel is usually burned as completely as possible with a small amount of air, and the resultant high-temperature gases are cooled by the remaining excess air in a specially designed mixing section, before reaching the turbine.

It is evident that any solid matter or ash that results from combustion must pass through the turbine blades. Although heavy fuel oils may contain incombustible ash, all coals do to a considerable extent and it is at the present impossible to contemplate the possibility of passing the products of coal combustion straight through a turbine. If the coal is to be burned as a fuel, the products must be passed through some kind of separator which will remove the solid abrasive and erosive particles. The achievement of a sufficiently fine degree of separation in a reasonable bulk and with reasonable pressure losses is probably the greatest problem in the direct firing of coal. As far as the actual combustion problem is concerned, it is closely akin to the problem of burning pulverized coal in a furnace, with the requirement of minimum bulk of firebox and the important difference that combustion must be carried out at some pressure between three and six atmospheres. There is no short cut to the solution of these problems, nor can they be evaded without introducing fresh problems.

One evasion of them that is being worked on is to convert the solid fuel to a combustible gas externally to the main furnace. Then the combustion problem is straightforward. The gas must, of course, be carefully filtered, but its volume is much less than that of the combustion products in the direct firing engine, and less bulky separators will be required. The complication is that, if the gas producer unit operates at low pressure, a compressor will be needed to force the gas into the combustion system, or, alternatively, the air supply to the producer may be pre-compressed and the whole producer run under pressure.

Apart from direct firing, another line of effort altogether lies in heating the air, in its passage from compressor to turbine, indirectly by heat transfer from an external source of heat. If this is done, it will be appreciated that, since no oxygen of the working air is consumed, the air may be used over and over again by cooling, as shown in Figure 3.

Such an arrangement is usually referred to as an 'aerodynamic turbine' and was first proposed and used by the Swiss engineers Ackeret and Keller, and the Escher Wyss Company. Since the hot gases which heat the working air do not pass

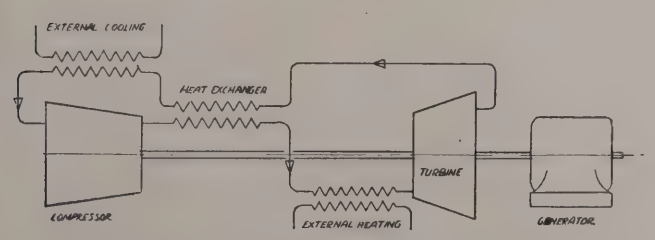

Figure 3.—Aerodynamic turbine.

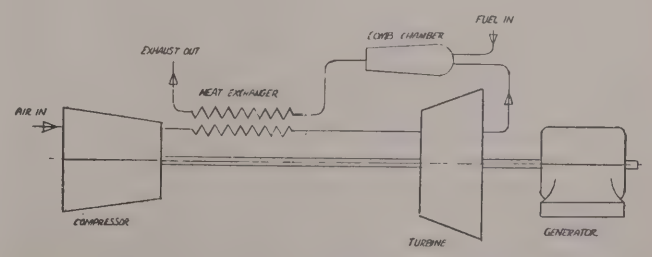

Figure 4.—Exhaust-heated gas turbine.

through the turbine, there is no *a priori* prohibition of coal as a fuel. Such an engine arrangement is well suited to large sizes and its rotational parts can be made small, since the absolute pressure of the working fluid may be made high. There are a number of arrangements of the so-called closed cycle, but they all have the same essential features. To burn coal it is necessary to build a coal-fired air heater. This may be regarded in principle as a boiler and furnace similar to a conventional unit except that there is no evaporation or steam generating space. All the heating surface is equivalent to superheater surface, but it runs hotter than conventional superheater tubes, even compared with a modern high-temperature steam generating unit.

Another possible solution, which in some ways is the simplest of all, is the so-called exhaust-heated engine. In this, the working air is heated by a coal-fired air heater, but the cycle is not closed and the hot exhaust from the turbine, instead of being cooled and re-cycled as in an open cycle engine, is used as the combustion air in the furnace, as shown in Figure 4.

To the best of the writer's knowledge, these four possible solutions are the only ones in which active work is being undertaken, and the progress being made will be briefly reviewed in the following section.

Present Position

(*a*) *Direct Firing of Pulverized Coal and Ash Separation*

In the past five years, an intensive programme of work has been carried out by the Locomotive Development Committee of Bituminous Coal Research, Inc., who have been developing a coal-fired turbine for use on the railroads.

Under the direction of Mr. John Yellott, the problems of coal preparation and feeding, combustion under pressure, and of ash separation have been vigorously pursued (1, 2).

As the result of more than 1,000 hours' experience operating an experimental gas turbine, much information has been gained on the performance of separators and on the erosion of turbine blades. Adequate protection can now be given to the blades. A 3,500 h.p. gas turbine suitable for locomotive application is now assembled and undergoing preliminary tests, using the equipment developed by the Locomotive Development Committee.

(1, 2) For references, see end of paper.

Tests made by the Federal Department of Mines and Technical Surveys have shown that the performance of the type of combustor used depends quite markedly upon the type of coal being burned. With low-volatile coals, the performance is not satisfactory and further work is required.

In the United Kingdom, the English Electric Company is building a 2,000 k.w. plant for experimental work (3).

(*b*) *Gas Producer Plants*

Work on gas producer units is proceeding both in the United Kingdom (3) and in the United States (4), but has not yet reached the stage of running an engine. Development of the pressurized gas producer suggests difficulties with respect to size and control.

(*c*) *Closed Cycle Units*

The first successful operation of a closed cycle engine was in 1939 (5) and it has been considerably developed since that time. The firm of John Brown & Co., Ltd., is actively engaged in the design and construction of engines based on this principle (6) and their first large unit, of 12,500 k.w., will soon be in operation. If we look back at Figure 3, it will be seen that all the heating of the working fluid is accomplished in an air heater. The main advantage of the closed cycle, with its large demand for cooling water, is that, by pressurizing the whole circuit, the unit may be made very compact. However, if this is done, it is desirable to pressurize the furnace also, so that the gas on both sides of the heater has a high density and, consequently, a high heat transfer. To do this it is necessary to use a furnace air-charging compressor, and far and away the most practical way of driving this is by a turbine, so that the air heater becomes a form of Velox boiler (7). If coal is to be used as fuel, the problems of the charging set are exactly the same as those met in the direct-fired engine.

If the furnace is not pressurized, there is no objection to the use of coal. The John Brown company has done work on the burning of slurries and peats. However, the only way of reducing the size of the air heater is by designing it to use a considerable amount of heat transfer by radiation, which may mean that best performance of a given design will only be obtained with one particular fuel.

(*d*) *Exhaust Heated Units*

The exhaust-heated unit (8 and 9) is an attempt to make a simple engine, in the smaller sizes, which can burn almost any solid fuel. The air heater uses a furnace at essentially atmospheric pressure and is therefore relatively bulky, a criticism which we levelled at the closed cycle in the paragraph above. However, by not closing the cycle, two considerable advantages emerge. Firstly, no cooling water is required, and secondly, the hot air exhausted from the turbine, or a portion thereof, may be used as the combustion air. The high degree of pre-heat is, of course, most beneficial for burning low-grade coals. The combustion problem is exactly that of a conventional forced draft furnace and there are no serious ash separation problems.

Design work on an exhaust-heated engine is being carried out by C. A. Parsons & Co., Ltd., in England, and some preliminary combustion and heat transfer tests have been made at the Fuel Research Station. On this continent, the Gas Dynamics Laboratory of McGill University, with the financial support and technical collaboration of the Department of Mines and Technical Surveys, is engaged in the construction of an experimental unit designed to secure the answers that must be obtained before one could embark with confidence upon the design of a locomotive engine.

Progress of Work at McGill

The work which we at McGill are doing under contract from the Department of Mines and Technical Surveys is to design and construct a small experimental coal burning turbine engine operating on the exhaust-heated cycle. The general arrangement of the plant is shown diagrammatically in Figure 5.

A two-stage centrifugal compressor and two-stage axial turbine are used and these are parts of a Rolls Royce 'Dart' aero engine. Two heat exchangers are used and the furnace fires directly into the first heat exchanger.

Figure 6 shows the coal handling system which is being installed. It will be noticed that the coal is not pulverized, but merely crushed.

A hydraulic brake is provided to absorb the power, and construction of the test bed is complete. The turbine part is presently installed for calibration, burning oil in its original combustion system. The coal handling system is now being erected. Detail design of the hot heat exchanger is being carried out by Canadian Vickers, Ltd., in collaboration with ourselves, and is well

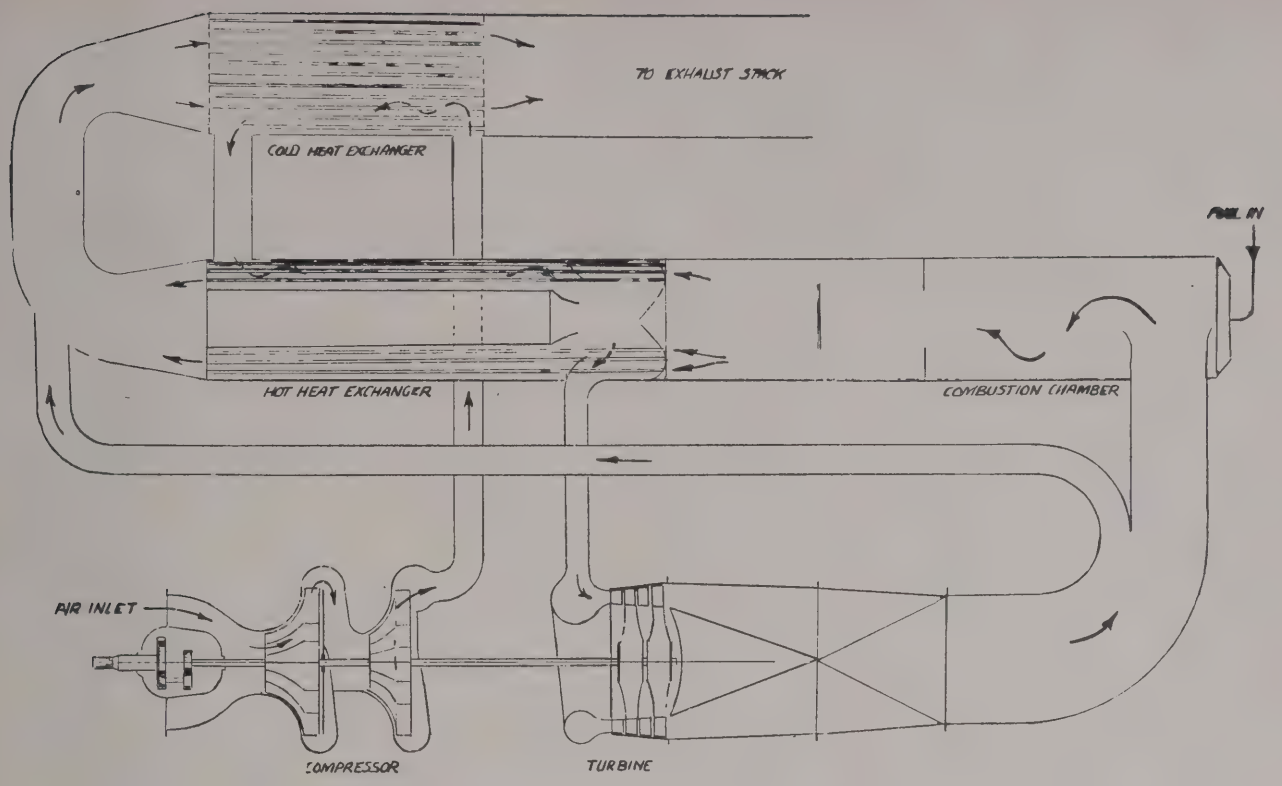

Figure 5.—Coal-burning gas turbine layout.

advanced. Nearly all of the design work is complete and the various components constructed. If not unduly delayed by shortages of material during construction, it looks as though the complete plant may be ready to run by about June of this year (1952).

In the meantime, there is much to be done. It is desired to use a cyclone type of slagging furnace, as this is well known to burn all types of low-grade coal with high efficiency. Development of the furnace is proceeding at Ottawa, under the direction of Dr. T. E. Warren, and good progress is being made. At the conclusion of the Ottawa tests, the furnace will be tested at McGill, using hot air from the 'Dart' engine, in order to prove the coal handling equipment and as a final check on furnace performance before assembling the complete unit.

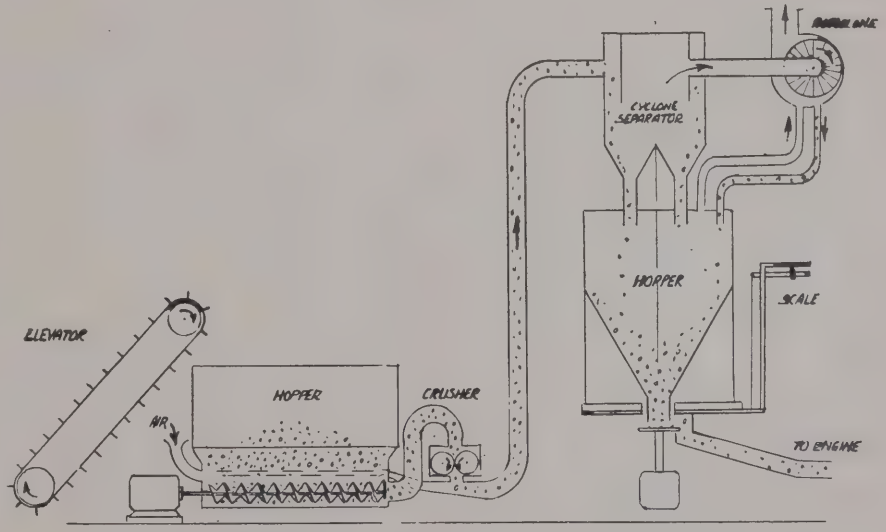

Figure 6.—Coal-handling equipment.

Prospects of the Coal Burning Turbine

We now turn from hard facts to peering into the future and consequently what follows is necessarily the personal opinion of the author. The question is, what can be expected of the coal burning turbine?

One can see two main uses for it, firstly on the railway and secondly for power generation. Considering the railway application, it seems indisputable to the writer that the gas turbine is going to receive wide adoption. The advantages commending it are:

(*a*) Its power increase in cold weather

(*b*) Its freedom from maintenance

(*c*) The fact that the gas turbine can be designed for mechanical, rather than electrical, transmission

(*d*) Lower fuel costs

It seems also to be indisputable to the writer that a gas turbine to burn coal is going to be more expensive than one designed to burn the lighter grades of oil. The relative cost of an engine using the lowest and heaviest residual fuel oils cannot be seen, but it is probable that it will be cheaper than the coal engine. Hence, in those regions where coal is sufficiently cheaper than oil to justify the extra cost, there will be a firm economic argument for the use of coal. In the west, however, there may not be sufficient difference, unless the cheaper western oils prove to be expensive to use in a gas turbine.

However, it is obvious that the correct answer to this question can only be given when a considerable amount of operating experience on both oil and coal fired turbine locomotives has been amassed. It looks as though there will be no coal turbine engine in service, other than that of the Locomotive Development Committee, within

the next four or five years. It is fruitless, therefore, to expect any early answer to this question. The profitable thing to do is to build coal burning engines of the various types and test them in service. There is no other answer and no short-cut.

In power station application, particularly in the smaller sizes, there would seem to be a field. The exhaust-heated engine is very simple, and the biggest trouble is the design of the high-temperature heat exchanger. The use of water cooling of some of the parts eases these problems considerably and, although not too easy to apply in a locomotive, in a power set it opens up new vistas. In any gas turbine, the bulk of the waste energy, amounting to 70 to 80 per cent of the calorific value of the fuel, is rejected in the exhaust gases. If only 25 per cent of this waste heat can be collected and put to use, very high overall thermal efficiencies are possible. In the exhaust-heated engine we already have what amounts to an air boiler and, by combining it with a water boiler, we can simultaneously solve some of the design problems of the hot exchanger and transfer some of the waste heat to steam, which can either be used to drive a steam turbine or for some process or heating work.

It is possible, then, to visualize a very compact power plant, using a combined air and water boiler, a gas turbine, and a steam turbine, that, particularly in the power range 1,000 to 6,000 k.w., should have a wide appeal, as its thermal efficiency would run between 40 and 45 per cent, a figure realisable by no existing power plant other than a mercury steam plant.

These, then, are the two duties for which the writer thinks the exhaust-heated coal burning turbine is well suited. It does not appear to be very practical for large sizes. For powers of 12,000 to 15,000 k.w., a closed-cycle coal burning engine is attractive and it appears probable that in the next few years such engines will be built. The open-cycle engine is inherently unsuitable for very large powers — this in spite of the fact that they have been built in sizes up to 27,000 k.w. — because the construction problems involved, owing to the need for large austenitic forgings and castings, make the unit exceedingly expensive. Since both the direct firing of coal and its prior gasification are methods essentially applicable only with open-cycle engines, it seems unlikely that these methods will be applied in the case of very large power outputs. For stationary purposes, one would expect such engines to appear in sizes up to 10,000 h.p., but the writer personally thinks that their inherent complication will militate against their adoption for locomotives, where maintenance is always such a problem.

There are still many problems requiring solution and much work to be done. The successful prosecution of the work now under way in the United States, Canada, and the United Kingdom will contribute largely to a more precise assessment of the future of the coal burning turbine.

Acknowledgments

The writer is indebted to Rolls Royce, Ltd., Derby, for making available the 'Dart' engine, on which the McGill testing programme is based; to the Locomotive Development Committee, for the loan of coal handling equipment, and to Mr. John Yellott, their Director of Research; and to Dr. Tom Warren, of the Fuels Division, Department of Mines and Technical Surveys, for many discussions and continual encouragement. He also wishes to thank Professor A. G. Lane, of McGill University, for preparing the illustrations for this report.

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Hydrogen Sulphide Corrosion Cracking of Steel

*By L. W. VOLLMER**

(*Annual Western Meeting, Calgary, Alta., October,* 1951)

(*Transactions, Volume LV,* 1952, pp. 79-85)

Introduction

DURING the past three years at least fifty strings of 9% nickel steel tubing have exhibited excellent resistance to the variably severe corrosive conditions typical of many of the sweet condensate wells in Mississippi, Louisiana, and Texas. This experience, and the characteristically outstanding performance of other nickel alloy steels under hydrogen sulphide corrosive conditions, dictated the selection of 9% nickel steel tubing for the second well drilled in the sour condensate Pincher Creek damaging effects of the Pincher Creek reservoir fluids on well equipfield of southwestern Alberta. The ment had been manifested in the discovery well by the rapid and not yet fully understood failure of API Grade N-80 tubing, stainless steel wire lines, and alloy steel fishing tools. Two very unexpected failures in the 9% nickel steel tubing, after only six days of service in production tests, instigated an extensive and continuing investigation to ascertain the probable cause of failure. This discussion presents some of the results obtained thus far which cast a little light on the perplexing behaviour of steels in hydrogen sulphide environments in general, and in sour condensate wells in particular.

History of Tubing Failures

Walter Marr well No. 1, in which the 9% nickel steel tubing failures occurred, produces from the Madison lime at the depth interval approximately 12,200 to 12,750 feet. The gas-oil ratio is about 26,000 and the gas contains 10.2 per cent hydrogen sulphide and 6.3 per cent carbon dioxide by volume. During six days of initial testing, the well was produced at an average rate of 7.5 million cubic feet per day, yielding about 300 barrels per day of distillate along with approximately the same volume of formation water. This water, containing some 37,500 p.p.m. dissolved solids, was high in bicarbonates and, of course, hydrogen sulphide. The bottom-hole pressure and temperature were 4,868 p.s.i. and 188°F., respectively, at 12,200 feet. The closed-in well-head pressure was 3,236 p.s.i.g. The well-head pressure at 7.5 million cubic feet per day flow rate averaged 1,350 p.s.i.g.

The 12,200-foot string of 9% nickel steel tubing (2⅞-in. o.d., 6.50 lb.) was set on a production packer at 12,186 feet. A 3,300-foot string of 5-inch casing was hung in the annulus of the tubing and the 7-inch oil string for circulation of hot Diesel oil (140 to 150°F.) down the 2⅞ in. to 5 in. annulus and up the 5 in. to 7 in. annulus. This raised the well-head flowing temperature to 96°F. and successfully prevented hydrate formation.

The first tubing failure occurred while the well was being killed for a cement plug-back to shut off water. Approximately 137 barrels of water at 50°F. had been pumped from a nearby lake into the annulus in the process of displacing Diesel oil (343 bbl.) when the well blew out through the water lines. It was found that the top tubing joint had parted 26 feet below the well-head and about 8 inches above the upset taper. Figures 1 and 2 show the failure.

While fishing, the tubing parted again at 1,201 feet under a load of 80,000 to 85,000 pounds at approximately the midpoint of the joint. Both halves of this failure are shown in Figure 3.

It is considered significant that, although large segments of both failures are brittle in appearance, there is positive plastic deformation of other segments so that the entire fractures cannot be classed as truly brittle.

After a successful plug-back to about 12,400 feet, the well was acidized and further tested through N-80 tubing with an internal plastic coating. The well was shut in for some 15 months, but recently limited production was resumed to test materials and methods of minimizing corrosion and preventing hydrate formation.

Analysis of Failures

The recovered 9% nickel steel tubing was examined at the well site by means of the Sonoscope. No further damage or outstandingly signifi-

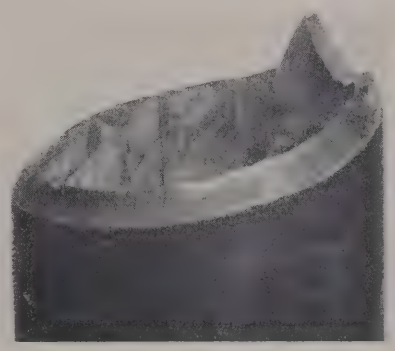

Figure 1.—One half of first tubing failure, 26 feet from surface.

Figure 2.—Other half of first failure.

*Director, Materials and Production Chemistry Division, Gulf Research and Development Company, Pitsburgh, Pa.

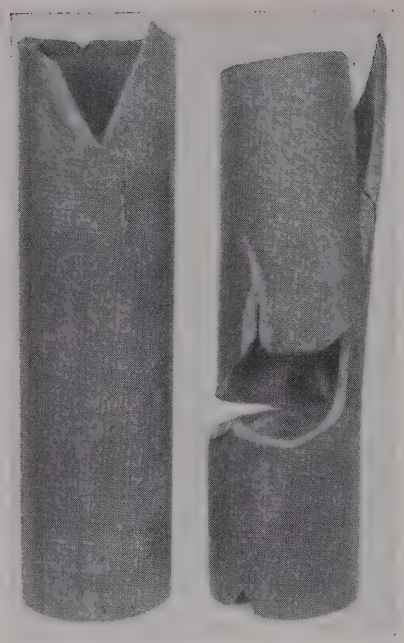

Figure 3.—Both ends of second failure which occurred at 1,201 feet from surface.

Table I.— Chemical and Mechanical Properties of Failed Tubing

	1st Failure	2nd Failure
Analysis—		
Carbon	0.095 per cent	0.094 per cent
Manganese	0.300 " "	0.360 " "
Phosphorus	0.025 " "	0.035 " "
Sulphur	0.027 " "	0.026 "
Silicon	0.230 " "	0.210 " "
Copper	0.210 " "	0.230 " "
Nickel	8.910 " "	8.990 " "
Chromium	0.070 " "	0.110 " "
Mechanical Properties—		
Yield point (p.s.i.)	101 600	121 300
Tensile strength (p.s.i.)	121 000	132 300
Elongation (% in 2 in.)	28.0	28.0
Brinell hardness	250	260

cant conditions were found. The tubing was remarkably free of general corrosion and, in fact, had the appearance of new, unused pipe.

Chemical and mechanical properties of the steel in both failures are listed in Table I; they are normal in all respects. The tubing had been heat-treated (normalized, 1,575 to 1,600°F. and tempered at 950 to 1,050°F.) to develop a required 90,000 p.s.i. minimum yield strength and 25 per cent minimum elongation (30 per cent desired).

Since the steel was not defective in any obvious respect, the unusual appearance of fractures (both ductile and nonductile) suggested the possibility that the failures may have been induced by triaxial loading resulting from tension weight of string), bursting stresses (contained gas), and complex stresses due to variable chilling by 50°F. water. Calculations involving combination of all maximum known stresses established that a torsional load sufficient to cause a twist of 45° in 30 feet, or one joint of pipe, would be required to develop a combined stress equal to the strength of the tubing. There was no conceivable source of such high twist, and it seems certain that the tubing failures were not purely mechanical.

It has been repeatedly suggested that the failures were caused by embrittlement, since the gas contained 10 per cent hydrogen sulphide. The writer and his associates (1), along with many other investigators, have shown that many steels exposed to hydrogen sulphide and water will lose considerable, if not all, ductility as a result of absorption of atomic hydrogen; but tensile and yield strengths remain essentially unchanged. In other words, steels embrittled by hydrogen do not fail under static tensile stresses until normal breaking loads are attained. Moreover, no significant change in ordinary mechanical properties was exhibited in tensile tests made on specimens of the tubing failures that had been immersed for short-time intervals in tap water saturated with hydrogen sulphide and carbon dioxide, and in well brine saturated with well gas.

It should not be overlooked, however, that absorption of atomic hydrogen may be a source of failure of steel under sulphide conditions by a process comparable to the well known blistering that occurs when some steels are acid pickled. Accumulation of molecular hydrogen at barriers within the steel (lattice imperfections, voids, inclusions, and sub-microscopic cracks) will produce high biaxial and triaxial stresses. These, plus normal service stresses, could conceivably induce brittle fracturing. There was no positive evidence that this had occurred in the 9% nickel steel tubing failures in the Marr well.

Since there was no possibility of fatigue, virtually the only remaining potential source of the tubing failures seemed to be stress-corrosion cracking. Stress-corrosion cracking is defined as spontaneous failure of metals by cracking under the combined action of corrosion and stress; the stress may be residual, applied, or both. The exact mechanism of the phenomenon is debatable. At least five theories (2) have been advanced, but it is suspected that no one theory applies in all cases of stress-corrosion cracking. It has been said that all metals can be made to fail by this process under some set of circumstances. Stress-corrosion cracking of metals is most common when the total stress is close to, or in, the plastic or yielding range.

Duplication of Failures

Initial attempts to duplicate the service failures were made on specimens cut from the second tubing failure (yield strength, 121,000 p.s.i.). Since oil well tubing is very often cold-straightened (plastically deformed) during fabrication, these specimens were loaded above the yield strength as single point beams and were then submerged in tap water containing hydrogen sulphide and carbon dioxide in a test cell at 5,000 p.s.i. and 190°F. (bottom-hole conditions). Partial pressures of hydrogen sulphide and carbon dioxide were adjusted to approximate and maintain well fluid concentrations The test period was six days (same as initial production test time), at the end of which time all specimens had failed, with brittle fractures in the zones of maximum stress (deformed regions).

Additional specimens from the second tubing failure were then single point beam loaded below the yield strength to maximum fiber stresses of 70,000, 90,000, and 110,000 p.s.i. and immersed in tap water saturated with hydrogen sulphide and carbon dioxide at atmospheric pressure and 90 to 100°F. The specimen loaded to 90,000 p.s.i. failed in 14 days at a point where the applied stress was only 63,000 p.s.i.; that loaded to 110,000 p.s.i. failed in 70 days where the applied stress was only 36,000 p.s.i. The specimen at 70,000 p.s.i. ultimately failed after 306 days at a point where the stress was 30,000 p.s.i. Figure 4 is a photomicrograph of the longitudinal section adjoining the complete fracture of this specimen, showing two stress-corrosion cracks. Figure 5 is a portion of one of these cracks at higher magnification and after etching the crack is predominantly transgranular. These er-

(1) For references, see end of paper.

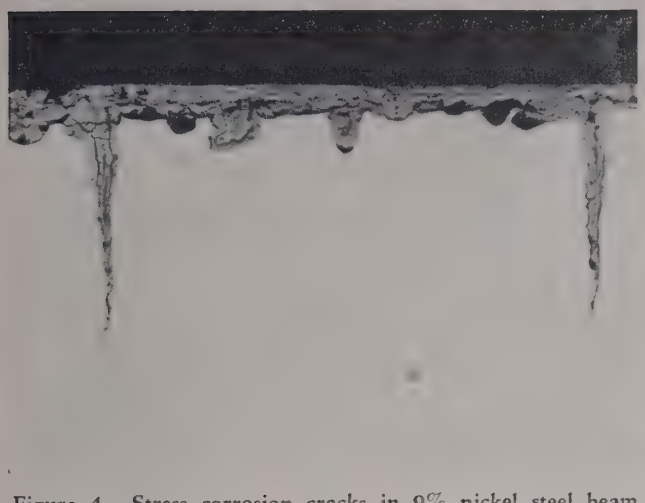

Figure 4.—Stress corrosion cracks in 9% nickel steel beam specimen (fiber stress 70,000 p.s.i.) after 306 days in corrodent (unetched).

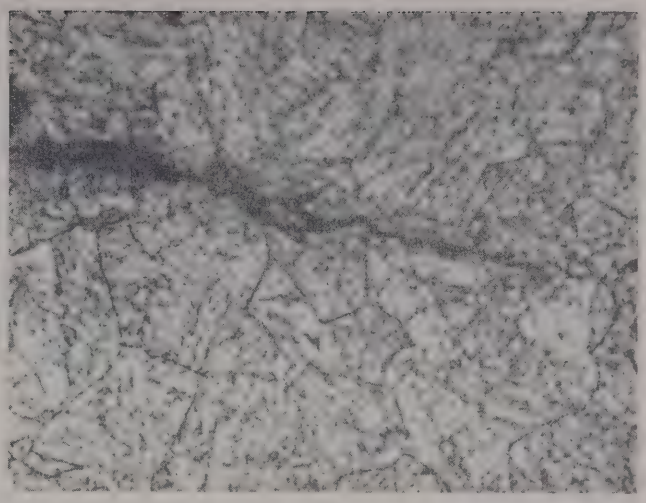

Figure 5.—Portion of one of cracks in Figure 4. It is predominantly transgranular. Nital. ×1,000.

ratic results suggested that failure had occurred in zones that had been stressed previously above the yield point by cold straightening the pipe from which the specimens were prepared.

A series of tests on 9% nickel and several other potentially usable alloys available as tubes, as well as API J-55 and N-80, was then undertaken to establish effect of pre-stressing above the yield point on susceptibility to stress-corrosion cracking. All alloys were re-heat treated to eliminate prior internal stresses. For the initial screening tests, specimens illustrated in Figure 6 were pre-stressed by loading in a tensile machine until the 1¼-inch length of uniform section in the centre was stressed 1,000 p.s.i. above the yield strength determined by stress-strain diagrams. The rectangular section of the specimen is graduated so that pre-stressing at the ends of the 4-inch mid-zone is only two-thirds of the yield strength. After pre-stressing, the specimens were ground full length to a uniform section (0.180 in. by 0.200 in.) and 2-point beam loaded

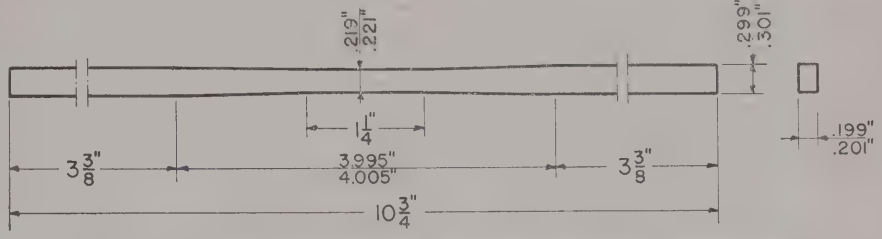

Figure 6.—Tensile specimen for pre-stressed corrosion cracking test.

Figure 7.—Method of applying bending load in stress-corrosion cracking tests.

in a fixture shown in Figure 7. The applied fiber stress is constant between the two points of loading 4 inches apart. Hence, the variably pre-stressed zone is subjected to uniform applied stress. A few tests were also made on externally loaded specimens that were re-heat treated but not pre-stressed. All specimens were immersed in tap water saturated with hydrogen sulphide and carbon dioxide at atmospheric pressure and room temperature. Additional laboratory and field tests are in progress on specimens of several steels pre-stressed midway between the yield and tensile strengths and to the ultimate strength of each metal.

TABLE II.— SUMMARY OF LABORATORY SULPHIDE CORROSION CRACKING TEST RESULTS

9% Ni steel. Corrodent in all cases tap water saturated with H_2S and CO_2 at atmospheric pressure and room temperature.

TEST NO.	YIELD STRENGTH (p.s.i.)	HARDNESS R_c	PRE-STRESSED*	APPLIED FIBER STRESS (p.s.i.)	TIME TO FAILURE (Days)	CLEANINGS (No. of)
1	121,300	28.0	Yes	120,000	1½ to 4	0
2	121,300	28.0	Yes	60,000	67	1
3	116,000	27.5	No	120,000	31 hours	0
4	116,000	27.5	No	70,000	202	3
5	116,400	27.0	Yes(*a*)	110,000	7 hours	0
6	92,000	24.0	Yes	90,000	202	4
7	75,000	20.0	Yes	73,000	(370) (*b*)	23

*Pre-stressed 1,000 p.s.i. above determined yield strength.
(a) Pre-stressed to ultimate strength (132,000 p.s.i.).
(b) No failure; still under test.

Laboratory Test Results

The more significant results for 9% nickel steel are summarized in Table II. With one exception (Test 7), all specimens of this steel failed sooner or later, depending primarily on the amount of pre-stressing above the yield strength and the magnitude of the applied fiber stress. Increasing hardness of the steel above Rockwell C24-26 and removal of corrosion products also appear to bear importantly on the exposure time to failure, which varied from seven hours to 202 days. Most of the failures, typical of which are those shown in Figure 8, were predominantly intergranular, with little or no evidence of pitting. Numerous small cracks parallel to the sides of the specimen, originating at and perpendicular to the main fracture surfaces (Figure 9), were present in many of the specimens. Short, longitudinal cracks were also found on top of some specimens (in the plane of maximum bending stresses); these, in association with the other cracks and the complete fracture, suggest a three-dimensional failure process.

The pre-stressed specimen in Test 2, Table II (applied fiber stress 60,000 p.s.i.) was removed from the test cell for examination after 66 days of immersion. Mechanical cleaning of a tightly adherent film of corrosion products exposed numerous very fine cracks. When this specimen was replaced in the test cell, failure occurred in 24 hours after a total immersion time of 67 days.

A non-pre-stressed beam specimen (116,000 p.s.i. yield strength) loaded to 60,000 p.s.i. fiber stress was removed from the test cell after 115 days continuous immersion without cleaning. Beneath a tightly adherent film of corrosion products numerous barely visible cracks were found at the base of etched lines or 'striae'. A microscopic examination showed that these etched lines were elongated pits approximately 0.001-inch deep with cracks at the base penetrating 0.001 inch or less. Figure 10 is typical of the condition found. The cracks are too shallow to be certain, but they appear to be transgranular for the most part.

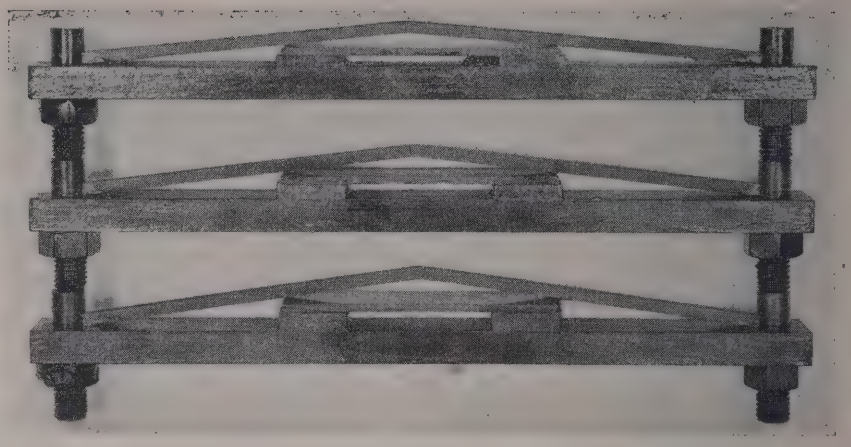

Figure 8.— 9% nickel steel stress-corrosion cracking failures.

The foregoing indicates that pre-stressing beyond the yield strength greatly increases the susceptibility of 9% nickel steel tubing (as ordinarily heat treated) to failure in the selected sulphide environment (Tests 1, 3, and 5). Plastic deformation is not essential to the failure process, however (Test 4). A film of corrosion products appears to afford partial protection to the steel in either condition when the applied stress is relatively low (Tests 2, 4, and 6); but after the film is mechanically removed or ruptured (for example, by running a bottom-hole pressure gage), failure may follow with disconcerting rapidity (Test 2). Also, it appears that the 'critical damaging stress'* is lower than 60,000 p.s.i. for 9% nickel steel as ordinarily processed for oil-well tubing, since all such specimens loaded to 60,000 p.s.i. or more eventually failed.

Significant results of sulphide corrosion cracking tests on eleven other steels and three non-ferrous alloys are summarized in Table III. Thus far, under laboratory test conditions, failures have been induced in a 5% nickel-1% chromium steel (Tests 10 and 11); 5% nickel-molybdenum steel (Test 15); and Types 322 and 410 stainless steels (Tests 21, 22, and 23). Apparently 5% nickel-1% chromium steel is less susceptible to failure than 9% nickel steel with comparable mechanical properties. Test 12 (no failure in 267 days) indicates that the 'critical damaging stress' for the form-

*'Critical damaging stress' is a relative term used for convenience; values of it cannot be employed for engineering purposes as they will vary with degree of plastic deformation of steel and corrosive environment.

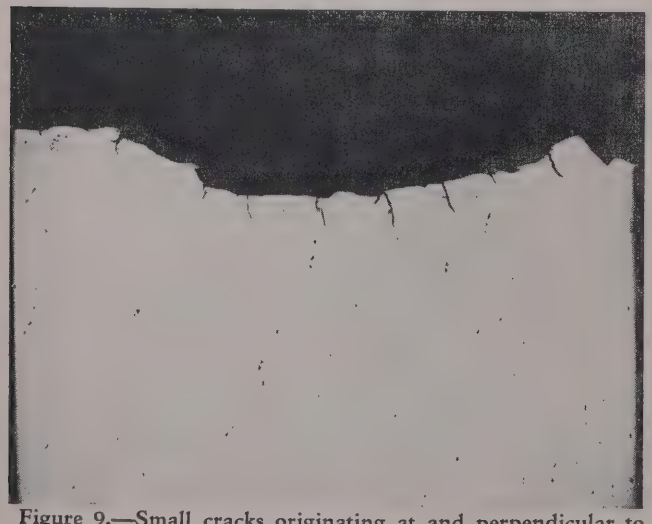

Figure 9.—Small cracks originating at and perpendicular to main fracture faces.

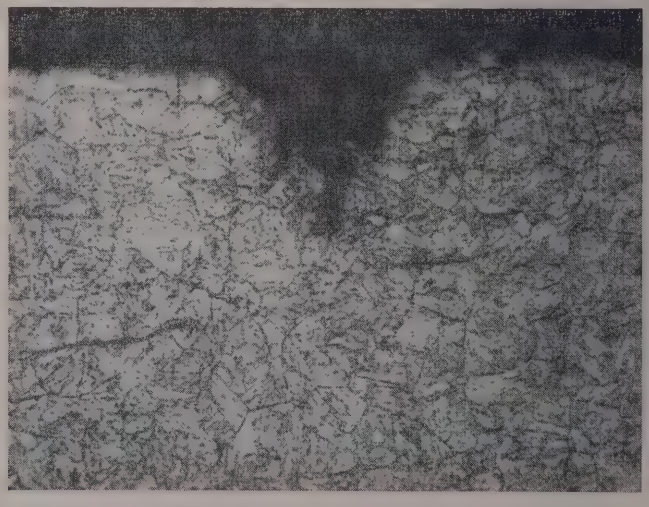

Figure 10.—Cracks starting at pits in stress-corrosion cracking specimen. Note small stringers of non-metallic inclusions. Nital etch. ×500.

TABLE III.— SUMMARY OF LABORATORY SULPHIDE CORROSION CRACKING TEST RESULTS

Corrodent: Tap water saturated with H_2S and CO_2 at atmospheric conditions.

ALLOY	TEST NO.	YIELD STRENGTH (p.s.i.)	HARDNESS R_c	PRESTRESS (p.s.i.)	APPLIED FIBER STRESS (p.s.i.)	TOTAL EXPOSURE (Days)	CLEANINGS (No. of)
9% Ni+0.30 Ti Steel	8	99,700	22.0	100,700	96,000	468	22
5% Ni Steel	9	62,000	15.0	63,000	60,000	521	21
5% Ni+1% Cr Steel	10	113,000	29.0	114,000	110,000	(3)(*a*)	0
	11	113,000	29.0	114,000	90,000	(3)(*a*)	0
	12	113,000	29.0	114,000	70,000	267	20
5% Ni+0.25% Mo. Steel	13	82,300	24.0	83,300	80,000	266	19
	14	81,700	24.0	101,400	80,000	57	7
	15	81,700	24.0	124,200	80,000	(23)(*a*)	3
9% Cr+1.0% Mo Steel	16	99,000	25.0	100,000	90,000	524	22
5% Cr + 0.55% Mo Steel	17	92,000	26.0	93,000	90,000	524	23
API Grade J-55	18	59,400	15.0	60,400	58,000	538	22
API Grade N-80	19	95,000	24.0	96,000	90,000	525	22
	20	95,000	24.0	118,000	90,000	236	18
Stainless 322	22	129,000	44.0	Not pre-stressed	120,000	(1)(*a*)	0
	21	129,000	44.0	1,000 p.s.i. above yield strength	120,000	(5)(*a*)	1
Stainless 410	23	105,000	25.0	"	100,000	(123)(*a*)	17
Stainless 416	24	90,000	24.0	"	85,000	179	24
Monel	25	105,000	24.0	"	100,000	179	24
K Monel	26	120,000	37.0	"	115,000	179	24
Inconel	27	40,000	80.0R_b	"	38,000	179	24

(*a*) Days to failure.

er steel is at least 70,000 p.s.i.; Test 4 on non-pre-stressed 9% nickel steel resulted in a failure at 70,000 p.s.i. applied fiber stress in 202 days.

The sensitizing effect of increasing amounts of plastic deformation is evident in the tests on 5% nickel-0.25% molybdenum steel under the same applied fiber stress (80,000 p.s.i.). A specimen of this steel pre-stressed 1,000 p.s.i. above the yield strength (Test 13) has not failed in 266 days, but pre-stressing to the tensile strength (Test 15) resulted in a failure in 23 days. A specimen pre-stressed midway between the yield and tensile strengths (Test 14) is intact after 57 days. The 'critical damaging stress' for this steel is apparently above 80,000 p.s.i.

Despite pre-stressing to its tensile strength, API Grade N-80 steel (average mechanical properties) has not failed in 236 days under 90,000 p.s.i. applied fiber stress. Although pre-stressed only 1,000 p.s.i above the yield strength, a specimen of 9% nickel steel with comparable yield strength (Test 6) failed at the same applied stress in 202 days.

The non-ferrous alloys tested have not shown any tendency to crack even when both hardness and applied fiber stress were high (Test 26).

With one exception (Test 15), all of the failures under laboratory test conditions have occurred in steels that had been heat treated to develop relatively high mechanical properties. This suggests that susceptibility to failure may be associated with the metallographic structure of the steel, and, therefore, might be minimized or eliminated by proper heat treating.

HEAT TREATING AND METALLOGRAPHIC STUDIES

After considering the heat treating and metallographic work on 9% nickel steel by Brophy and Miller (3), it was postulated that susceptibility of 9% nickel steel to failure under hydrogen sulphide conditions might be due to untempered martensite resulting from decomposition of retained austenite upon cooling after tempering, or when the steel is plastically deformed. Untempered martensite is highly strained and, therefore, might be anodic to adjacent metal. If the untempered martensite were present as a more or less continuous network, it could be a source of stress-corrosion cracking.

A series of heat treating experiments and metallographic studies on 9% nickel steel were undertaken in an attempt to prove the theory. Figure 11 is the etched structure at 2,000 diameters magnification of 9% nickel steel after annealing, followed by tempering for half an hour at 800°F. The areas of dark grey constituent were found to have a microhardness equivalent to 500-550 Brinell, indicating that the constituent is probably martensite. Figure 12 represents the structure near the first failure in the 9% nickel steel tubing. The dark grey constituent is present at some of the grain boundaries. Appropriate etching has proved that this constituent is not a carbide.

Only one heat treatment was found which appears to eliminate the dark etching constituent in, and the susceptibility to sulphide corrosion cracking of, 9% nickel steel. The structure of this steel after double normalization at 1,650° and 1,450°F., followed by tempering at 1,100°F. for two hours, is shown in Figure 13.

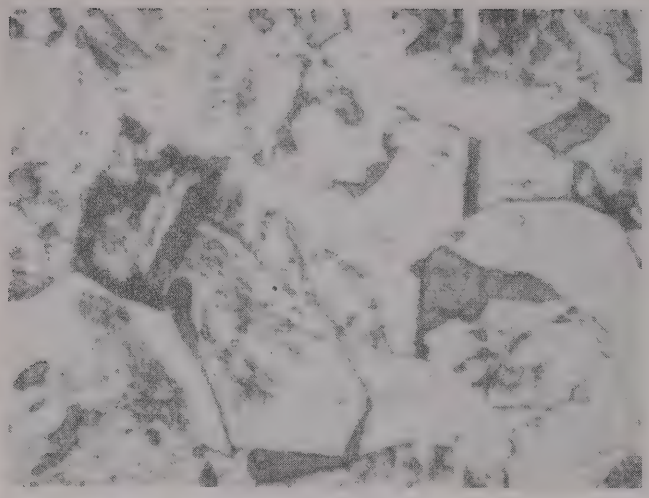

Figure 11.—9% nickel steel structure after annealing followed by tempering at 800°F. Hardness of dark constituent is 500-550 Brinell. Nital etch. ×2,000.

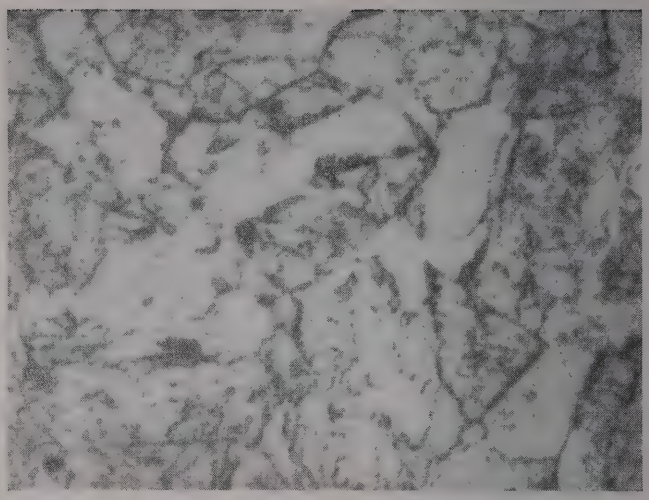

Figure 12.—Structure near first tubing failure, showing dark constituent at some grain boundaries. Nital etch. ×2,000.

A pre-stressed beam specimen of 9% nickel steel so heat-treated has been immersed in the laboratory corrodent for 370 days (Test 7, Table II), during which time products of corrosion were removed 23 times. Thus far, there are no indications that sulphide corrosion cracking is occurring. Unfortunately, the successful heat treatment is impracticable from a tubing fabrication standpoint, and the mechanical properties of the steel are undesirably low for high-pressure well service.

The effects of alloy 'stabilizing' additions have been explored only to a very limited extent, but there are indications that titanium and possibly molybdenum may be beneficial in reducing the sensitivity of some steels to the cracking phenomenon. It will be noted in Table III (Test 8) that a pre-stressed beam specimen of 9% nickel steel with 0.30 per cent titanium has been under test for 468 days without failure under an applied fiber stress of 96.000 p.s.i. The structure of this steel is shown in Figure 14. Carbides are present but the dark etching constituent is absent. It seems probable that the titanium combines with the carbon of the steel to form titanium carbide, thereby minimizing or preventing austenite retention and thus avoiding the possibility of the subsequent formation of untempered martensite. Possibly, molybdenum in relatively large amount will serve in the same capacity. This appears to be indicated by the resistance of 9% chromium-1.0% molybdenum steel to sulphide corrosion cracking (Test 16), in contrast with the failure of 12% chromium steel (Type 410) in 123 days (Test 23).

The absence of any known failure of 9% nickel steel tubing in sweet condensate service indicates that hydrogen sulphide is essential to the corrosion cracking process. This is confirmed in part by stress-corrosion tests on beam specimens from the second 9% nickel steel tubing failure in the Marr well. No failures have occurred after 454 days of immersion in tap water saturated with carbon dioxide, despite pre-stressing and beam loading to 100,000 and 120,000 p.s.i. Considerable corrosion in the 4-inch zone of high stress has occurred, as shown in Figure 15.

A comprehensive pilot production test was started at the Marr well in March, 1951. The programme, which is nearing completion, includes extensive materials studies under actual operating conditions, in which pre-stressed beam specimens, tube sections, and corrosion coupons of a large group of steels and non-ferrous alloys are being utilized. Detailed results of this work will be the subject of future reports. All of the failures of beam specimens that have occurred in the laboratory studies discussed herein have been duplicated under field conditions. In addition, failures have occurred in the field in beam specimens of 5% chrome steel, which so far has suc-

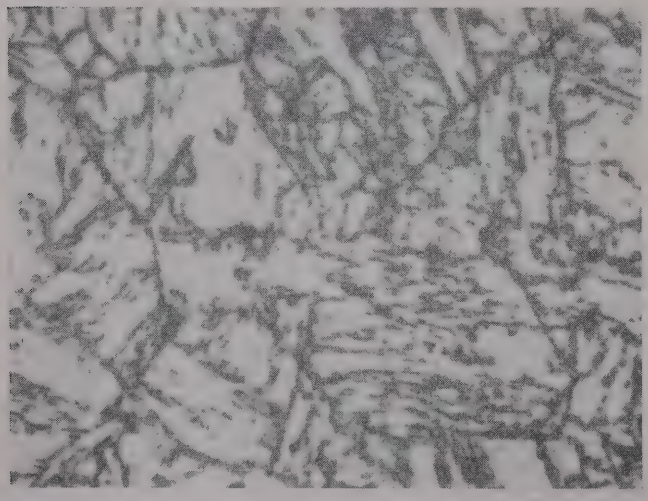

Figure 13.—9% nickel steel double normalized 1,650-1,450°F., tempered two hours at 1,100°F. Dark etching constituent is absent. Nital etch.

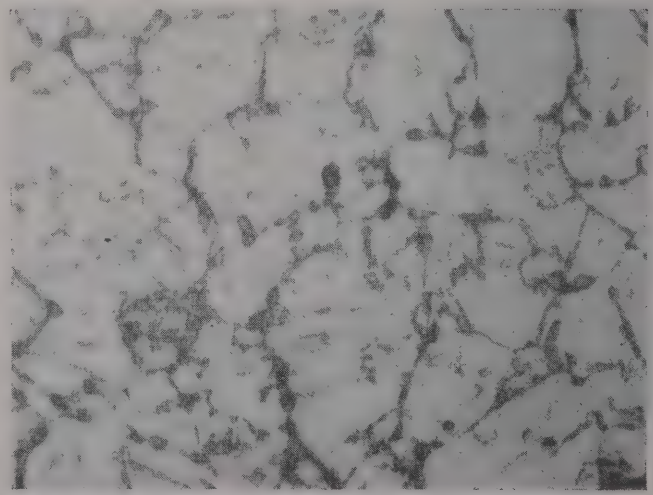

Figure 14.—Structure of 9% nickel steel with 0.300% titanium. Dark constituent is absent. Nital etch.

cessfully withstood laboratory test conditions. This, and the fact that in some instances failures were more rapid in the field, indicates that the field operating conditions, involving higher pressures and different water, are more severe than the laboratory conditions selected for screening purposes.

Summary

This investigation obviously is not completed, but the developments thus far seem to justify the following observations:

(1) The failures of 9% nickel steel tubing at Pincher Creek were initiated by stress-corrosion cracking.

(2) Susceptibility to failure in hydrogen sulphide environments is not limited to 9% nickel steel. There is evidence to indicate that most steels heat-treated to produce mechanical properties in or above a critical range represented approximately by Rockwell hardness C24 to C26 may be rendered susceptible to the failure process.

(3) Plastic deformation greatly increases susceptibility of steel to the failure process, but it is not an essential condition if the protective film of corrosion products is ruptured or removed by some means, and the hardness of the steel is in or above the critical range Rockwell C24 to C26.

Figure 15.—9% nickel steel beam specimens after 272 days; tap water saturated with carbon dioxide. Specimen loaded to 120,000 p.s.i. shows more corrosion in 4-in. high-stress zone.

(4) The mechanism of hydrogen sulphide corrosion cracking of steel has not been proved, but there is evidence that a metallographic structural phase as a network may be the sensitizing factor.

(5) Embrittlement of steel by exposure to hydrogen sulphide is not considered a primary cause of failure, but it may be an important contributory factor.

(6) It may be possible to reduce susceptibility to the failure process by composition changes in steel.

(7) There is no evidence thus far that 9% nickel steel will stress-corrosion crack in sweet condensate well service.

The author wishes to acknowledge the invaluable aid of his associates: A. J. Teplitz, C. N. Bowers, A. E. Wiehe, W. J. McGuire, J. K. Rodgers, and R. R. Ellis.

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Recent Research on Saskatchewan Bentonites*

By R. V. TOMKINS †

(*Annual Western Meeting, Calgary, Alta., October,* 1951)

(*Transactions, Volume LV,* 1952, pp. 86-88)

Introduction

IN THE past fifteen years, bentonite production in the United States has increased very rapidly. From 150,000 tons in 1934, it rose to 400,000 tons in 1942 and to a record 920,000 tons in 1948. Occurrences of bentonite in western Canada were described in the early part of the century by Ries and Keele (3), Ells (2), Spence (4), and other officers of the Dominion Department of Mines, and, in Saskatchewan, Professor W. G. Worcester (6), of the University of Saskatchewan has explored and carried out tests on deposits of the material at various times during the past thirty years. Nevertheless, Canadian developments have not kept pace with those in the United States. Exact figures for total Canadian production are not released, but it is probably under 10,000 tons annually.

There are a great number of uses for bentonite, but in the United States three alone account for 90 per cent of the consumption. These are for drilling gel, foundry sand bond, and oil decolorizer. The tremendous oil exploration programme on the Canadian prairies has increased the possibility of an adequate domestic market for drilling gel. Therefore, in Saskatchewan, our efforts have been directed mainly to the production of a good drilling gel, with a minor part of the project devoted to activation of bentonites for oil decolorizing.

Occurrences and Reserves

The main known occurrences of bentonite in Saskatchewan are in Frenchman River valley, in the Rockglen, St. Victor, and Pickthall areas of the south-central part of the Province, and in the Pelly district of eastern Saskatchewan. The large deposit near Knollys, in Frenchman River valley, occurs in the Battle formation of late Upper Cretaceous age and is estimated by Worcester (6) to contain over two million tons of bentonite. The Pelly occurrences are in the Pembina member of the Vermillion River formation of earlier Upper Cretaceous age. (Manitoba bentonite production is from the Pembina). We have drilled a number of core holes in this deposit, proving that the tonnage available is considerable, but, because the beds are folded, no satisfactory estimate of reserves has been made. The south-central deposits are all of Tertiary age, in the Ravenscrag formation. The Rockglen deposit contains over a million tons of bentonite according to Professor Worcester. In the St. Victor-Pickthall area, we have proven about 200,000 tons by drilling, and bentonite outcrops at numerous other points along the valley of Twelve Mile Lake, from west of Pickthall to Willowbunch, a distance of fifteen miles.

Structure and Types of Bentonite

Bentonite is composed mainly of montmorillonite, a mineral that was formed by alteration of volcanic dust laid down in waters of earlier eras. This mineral is a substituted form of pyrophyllite, $4SiO_2.Al_2O_3.2H_2O$, which is an electrically neutral molecule and, structurally, is considered to be a very thin plate consisting of a gibbsite ($Al(OH)_3$) layer with a silicon tetrahedron (SiO_4) layer on either side. Montmorillonite has the same general structure as pyrophyllite, but in it some of the aluminum atoms are replaced by magnesium and iron, leaving the molecule electrically unbalanced, carrying a negative charge. To satisfy this charge various positive ions, usually sodium or calcium, are attracted and held to the molecule. The sodium ions on the surface of the clay particles have a strong attraction for water and are thus responsible for the absorption by the bentonite of large quantities of water, with consequent swelling. Water penetrates between adjacent layers of oxygen atoms in the montmorillonite crystal, separating the individual plates. Calcium ions, however, do not hydrate, and hence, where calcium is satisfying the negative charge of the clay molecule, no swelling occurs. Also, as calcium is divalent, one bond may be connected to each of two adjacent montmorillonite plates, preventing them from separating, and preventing the water molecules from reaching any sodium atoms which may be available for hydration on either of these plates.

Sodium bentonites are used for drilling gel and for foundry sand bond, and usually need only be dried and pulverized before use. Bentonites used for oil-decolorization must be subjected to violent treatment with acid, which dissolves a considerable part of the clay. Consequently, the type and degree of internal substitution, as well as the nature of the adsorbed ions, determine the suitability of any particular bentonite for this purpose.

Activated Bentonite for Decolorizing Oils

We have continued the testing of Saskatchewan bentonites initiated by Professor Worcester, which consists of activating* the sample according

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†Industrial Research Engineer, Saskatchewan Department of Natural Resources.

(3) For references, see end of paper.

*Activation: The bentonite was dried at 220° F.; ground 90% through 200 mesh; refluxed for two hours with 2.5 times its weight of 18% hydrochloric acid; filtered, washed, and dried at 110°F.; and finally ground 90% through 200 mesh.

Decolorization Test: The desired percentage of clay was added to the un-decolorized Marvelube 30 oil and the temperature raised to 300°F. in 15 minutes with constant stirring. The mixture was then vacuum filtered through two No. 5 Whatman papers and the oil compared with Lovibond colour standards.

to standard procedure and comparing its decolorizing with that of a commercial decolorizing clay. Bentonite from the Rockglen deposit proved as good as the standard, while that from Pelly was found to be somewhat superior.

Drilling Gel

In the rotary drilling of oil-wells, drilling fluid or 'mud' is pumped continually down the hollow drill stem, and returns to the surface through the annulus formed by the drill pipe and the wall of the hole. This mud serves several important functions: it lubricates the bit, carries cuttings to the surface, holds cuttings in suspension when circulation is stopped, and, most important, forms an almost impervious cake on the wall of the hole, preventing passage of water to the formations being drilled, and preventing caving of the walls of the hole. The greater portion of the solids in this mud is usually clay picked up when drilling through shale formations. Bentonite is added to give the mud sufficient viscosity, to obtain the desired gelling characteristics, and to maintain a minimum loss of fluid to the formation and a minimum thickness of mud on the wall of the hole.

Although development of high viscosity is no longer the main function of the bentonite, nevertheless, the bentonite must produce high viscosity suspensions at low concentration, or it cannot be economically used. This property is evaluated in the drilling trade as 'yield', *i.e.*, the number of barrels of mud of 15 centipoise viscosity that can be prepared from one ton of bentonite. Gels presently in use yield 90 to 110 barrels per ton. Preliminary tests of Saskatchewan bentonite samples showed that none would yield over 35 barrels per ton. Our efforts were then directed to modifying a bentonite, so that a higher yield might be obtained. X-ray diffraction photographs taken and interpreted by Mr. E. C. Bailey, of the Chemistry Department of the University of Saskatchewan, showed that only samples from the St. Victor-Pickthall area gave strong montmorillonite patterns comparable to those of American swelling bentonites. As these samples also had produced the highest yields in preliminary tests, all further work was concentrated on samples from this area. Chemical analysis of clay from near Pickthall is compared in Table I with an analysis by the United States Bureau of Mines of a high-grade swelling bentonite from Belle Fourche, South Dakota (5).

Table I.— Analyses of Saskatchewan and South Dakota Bentonites

Constituent	South Dakota	Saskatchewan
SiO_2	60.6	64.1
$Al_2O_3+TiO_2$	23.4	17.5
Fe_2O_3	3.9	5.7
CaO	0.6	1.6
MgO	2.4	2.2
Alkalis (as Na_2O)	4.7	1.3
Ignition loss	2.8	6.0
Total	98.4	98.4

The Saskatchewan clay contains more calcium and less sodium than the Dakota clay, and, to make it swell more, the calcium ions must be removed and replaced with sodium. The obvious reagent to effect this change is sodium carbonate, which forms insoluble calcium carbonate with the calcium ions and supplies sufficient sodium to replace the calcium. By adding sodium carbonate to water in which the bentonite was dispersed, the yield was increased to between 50 and 55 barrels per ton. In order to raise the concentration of sodium carbonate, without increasing the ratio of sodium carbonate to clay, and to force the reaction nearer to completion, dry soda ash was added to the wet bentonite as mined, containing about 25 per cent moisture. The bentonite was then blended in a Simpson intensive mixer, dried, and pulverized. This treated clay proved much better than the original, yielding 85 barrels per ton in distilled water, and 100 barrels in hard water.

In the course of experiments to determine the optimum chemical addition, anomalous results were noticed which showed that factors other than the amount of chemical were operative. The moisture content of the clay as added to the mixer was found to have a lower limit below which beneficiation became less effective, and an upper limit beyond which the clay became too stiff and moved around the machine in large lumps without mixing properly. We also found that, if the clay became too dry, it did not re-hydrate properly. This has been reported by other workers and is believed due to strong cohesive forces developing as the water is removed and the montmorillonite plates come very close together. Sodium carbonate addition of about 2 per cent of the weight of dry clay was found to be best for this particular bentonite. Smaller amounts apparently do not replace all the calcium, while in greater percentage the excess soda ash becomes a deflocculant and begins to dehydrate the clay. Either effect results in lower yield.

The most interesting factor affecting the yield of the bentonite is the 'intensity of mixing'. This is a function of the duration of mixing for a given size of batch in a particular machine. That mechanical action alone was partly responsible for the improved yield was proved by running a batch without chemical addition. Samples were taken after various periods, dried, and pulverized, and from these 8 per cent suspensions were prepared. The viscosities of the suspensions are given in Table II.

Table II.— Effect of Duration of Mix-Mulling on Viscosity

13.3 lb. of wet bentonite (25% moisture) in 30-inch mixer. Samples dried and pulverized; suspensions 8 % by weight.

Duration (minutes)	Viscosity, Stormer (at 600 r.p.m.)
0	7
5	14
10	18
20	19
30	19

Our explanation of this effect of mix-mulling is that, under the heavy mullers of the Simpson mixer, the clay particles are squeezed many times, the number depending on the duration of the treatment, and this slides the montmorillonite plates around with consequent breaking of some of the calcium linkages. When a suspension is subsequently prepared, water is able to penetrate between the plates and hydrate any sodium ions available. When sodium carbonate is present, the broken calcium linkages are immediately inactivated by carbonate ions, and the sodium ions become attached to the free bond on the clay particles, effectively converting the clay from a calcium to a sodium bentonite.

Table III is a compilation of data obtained with the prepared clay, using standard procedures of the American Petroleum Institute (1) for laboratory evaluation of drilling clays.

Table IV was prepared from curves of yield *vs.* viscosity to show the yield obtained at various viscosities from clay suspended in distilled and in hard water. It will be noted that higher yields were obtained

TABLE III.—PROPERTIES OF SASKATCHEWAN BENTONITE SUSPENSIONS

(All tests performed according to A.P.I. Code 29, procedure for laboratory evaluation of drilling mud material)

WATER USED	BENTONITE (% by weight)	BARRELS PER TON	STORMER VISCOSITY (centipoise)	pH BECKMANN	GELS, STORMER		FILTER TEST 30 min. at 100 p.s.i.	
					Initial	10 min.	FILTRATE, cc.	CAKE 32nds.
Distilled	5	111	6	9.2	0	20	10.9	1
	6	92	11	9.2	0	35	10.8	1
	7	78	20	9.2	5	60	9.8	1+
	8	68	33	9.1	20	80	8.6	1.5
Hard*	5	111	11	8.5	10	33	13.1	2
	6	92	19	8.5	25	50	11.7	2
	7	78	29	8.5	45	77	10.8	2
	8	68	53	8.5	60	105	10.8	2+

*Contained 1,100 p.p.m. total solids; 183 p.p.m. calcium; 94 p.p.m. magnesium. Total hardness as $CaCO_3$: 850 p.p.m.

when hard water was used whereas the commercial gels showed a marked decrease in yield with increase in the calcium content of the water used for mixing. As most water used in western Canada for preparing mud is hard, this clay offers a definite advantage. Except that the yield is slightly lower in very soft water, the Saskatchewan bentonite compares favourably with commercial gels. The cake formed in the filter test is thin, and the water loss is low. Further laboratory tests proved it to be less easily flocculated by calcium sulphate or sodium chloride than a standard gel used for comparison.

After proving the effectiveness of this bentonite in laboratory tests, we adapted a nine-foot dry pan as a mix muller and produced a car-load of prepared bentonite for field tests. It is being used at the present time in two Saskatchewan wildcat wells and was used successfully to drill to the anhydrite in another. It has also ben used for shot-hole drilling by seismic exploration companies, who have expressed satisfaction with its performance.

TABLE IV.—YIELD OF MUD OF VARIOUS VISCOSITIES FROM SASKATCHEWAN BENTONITE
(Barrels per ton of clay)

STORMER VISCOSITY (Centipoise)	IN DISTILLED WATER	IN HARD WATER*
10	95	114
15	85	100
20	79	90
30	71	77
40	65	71
50	59	67

*See footnote to Table III.

SUMMARY

There are in Saskatchewan valuable deposits of bentonite awaiting exploitation as adequate markets become available. When properly activated, bentonites from deposits near Rockglen and Pelly produce oil-decolorizing clays which are as good or better than clays now used for that purpose. Bentonites from the St. Victor-Pickthall area can be modified by a relatively simple process to produce a good drilling gel. Adequate reserves for commercial operations have been proven by core drilling, or by measurement of outcrops.

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Report of Council

For the Year Ended December 31st, 1951

(*Transactions, Volume LV,* 1952, pp. 89-97)

To the Membership of the Institute
Gentlemen:

THE MINERAL INDUSTRY OF CANADA IN 1951

CONTINUANCE of the remarkable, successive increases in the value of Canada's mineral production, chief basis of the nation's post-war expansion, was one of the highlights of Canadian development in 1951, when, according to the preliminary estimate issued by the Dominion Bureau of Statistics and reported in the Bulletin for February, 1952, mineral output for the year reached a new high of $1,228 millions. This marked an increase of 17.5 per cent over the previous year, which was the first in which the billion dollar mark had been reached, and an increase of 143.7 per cent over 1946, when production was valued at approximately $503 millions.

While higher prices for the principal base metals accounted for a large part of the 1951 increase, quantity gains were made by copper, nickel, zinc, asbestos, lead, cement, and strikingly important, by petroleum, the value of which exceeded for the first time that of coal production. Gold was an exception to the general trend, being lower in both quantity and value than in 1950. For the first time, nickel came within $9 millions of surpassing gold as the leading mineral product.

Metals and Fuels

The value of all metals, estimated at $733 millions, was 18.7 per cent more than in the previous year. Nickel advanced 11 per cent in volume and 34 per cent in value. Copper was up 2.4 per cent in quantity and 21 per cent in value, zinc rose 6.6 per cent in quantity and 35 per cent in value, and although the volume of lead produced was lower by 8 per cent, its value increased by 17 per cent. The tonnage of iron ore was 31 per cent higher than in 1950. *Contra,* a lower price for gold — $36.85 per troy ounce compared with $38.05 in the previous year — combined with lower volume, resulted in a decrease of $10 millions in the value of gold produced.

A gain of 18 per cent in the value of all mineral fuels, bringing the total to $238 millions, was due almost entirely to the substantial increase in production of crude petroleum in Alberta. The volume of this increasingly important product reached 48 million barrels, compared with 29 million barrels the year preceding, and the value increased 43 per cent to $121 millions. Coal production, estimated at close to 19 million tons valued at $110 millions, was slightly lower than in 1950.

Industrial Minerals

Substantial gains were made also in the broad field of industrial minerals, including structural materials and all non-metallics except fuels. The overall increase for these groups was $30 millions, or 13.2 per cent; and the total value for the year, $257 millions.

Asbestos, the leader, scored an advance of 10 per cent in tonnage and 20 per cent in value to 967,000 tons worth $78.8 millions. Sand and gravel came next with a production value of $43 millions, and the increasing demand for structural materials was also reflected in the quantity and value of cement, clay products, and lime produced.

Following asbestos in the group of non-metallic minerals were such items as salt, fluorspar, magnesitic dolomite and brucite, nepheline syenite, quartz, sodium sulphate, and sulphur in the form of pyrites and sulphuric acid, all of which made impressive gains. Due entirely to a change in the method of compilation, gypsum production, which was substantially the same as in 1950, was reported in lower figures.

Looking Forward

What of the future? Irrespective of abnormally high prices for all minerals and metals except gold, stimulated in part by rearmament and in part by natural scarcities and inflation, the momentum of Canada's rising mineral production is too dominant to cease abruptly, even if, as may be hoped, more tranquil times ensue. Moreover, production appears to be slowly but surely adjusting itself to the impetus of new, large-scale investment in our abundant, latent resources, and the process of adjustment should continue to influence the upward trend.

Indeed, our still young nation at long last appears to be fast moving in the direction of fulfillment of that oft-mentioned prediction of Laurier's that *the Twentieth Century belongs to Canada,* and we of the Institute should be proud that expansion within the industry we foster is chiefly responsible for making this possible.

From silver to gold and base metals, from coal to ample sources of crude oil, from asbestos to sulphur and a host of other industrial minerals, we are advancing toward greater industrial opportunity through iron and steel, through greater utility of natural gas, through water power and waterways, all of which are bringing in their wake both secondary industries and a more united and prosperous Canada.

The dominant position of the mineral industry in overall Canadian expansion confronts the Institute— particularly its younger members and future leaders — with an unusual challenge. The impressive story of opportunity only needs to be told — and *sold* — to instill in Canadian youth an increasing desire to seek careers within the mineral industry. And this constitutes our greatest problem.

We continue to view with grave concern the adverse positions of the coal and gold industries, but even here there are rays of hope. Despite stiffer competition from other fuels and problems of distribution, coal productoin is holding its own, and its ultimate expansion should accompany the industrialization of the Maritime and Prairie provinces, a

corollary of iron ore and steel developments. There is increasing evidence, also, that the need to raise the basic price of gold in line with production costs and deflated currencies will, in time, be recognized by action.

THE INSTITUTE

The Institute also had a busy year. While fewer in number, major meetings were well attended and conducted, but the basic strength of the organization was best shown in the effective work of a growing number of active Branches, each of which contributed substantially to the success of the whole.

Another index of general health and interest was the remarkable response of the voting membership, mentioned elsewhere in this Report, to the proposal of Council that annual subscription rates be increased so that present services might be maintained without incurring further deficits. We recognize that adoption of this proposal by an overwhelming majority also created an obligation to redouble efforts to improve and, where possible, to expand services.

Major Meetings

The Fifty-Third Annual General Meeting, sponsored by the Quebec City and Thetford Mines Branches, was held in Quebec City on April 9th to 11th. As fully reported in the May, June, and July Bulletins, this excellent convention, attended by 1,112 members, ladies, and guests, was preceded by sessions on *Education for the Mineral Industry* and followed by an instructive two-day tour of the asbestos mines in the Eastern Townships of Quebec. Principal highlight of the meeting itself was the well organized symposium on *The Future of Iron Ore in Canada.*

In conveying our thanks to the General Committee of the sponsoring Branches, headed by Mr. Eugene Larochelle, we are not unmindful of the personal contributions made toward the success of the meeting by Dr. A. O. Dufresne, then President, by the Ladies' Committee headed by Mrs. R. H. Taschereau, and by the Technical Programme Committee headed by Mr. L. W. Bladon. We also commend the Quebec City Branch for remitting to the Institute $1,000 as a reserve to help guarantee Institute Branches against losses in conducting future Annual General Meetings.

As reported in the November Bulletin, the *Thirty-Third Annual Western Meeting,* also largely attended, was held in Calgary on October 3rd to 5th, in conjunction with the *Second Annual Technical Meeting of the Petroleum and Natural Gas Division.* Although the Calgary Branch and the Petroleum Division were the chief sponsors, all the Technical Divisions participated in the programme, and both the Coal Division and the B.C. Section co-operated by conducting special luncheons.

All concerned with the success of a western convention that vied in importance with the Annual General Meeting deserve our warmest thanks and praise. For their zeal and efficiency, we particularly mention Messrs. W. D. C. Mackenzie, Chairman of the General Committee, D. W. M. Ross, Chairman of the Calgary Branch, J. B. Webb, Chairman of the Petroleum and Natural Gas Division, and J. W. Young, Chairman of the Programme Committee.

On January 21st to 23rd, 1952, a *General Meeting* of the Institute was held in Ottawa. Much of the organizational work for the meeting was, of course, completed in the latter part of 1951, and the general proceedings have already been reported in the Bulletin for February, 1952. Further comment on the meeting is appropriately reserved for the 1952 Report of Council.

Institute Honours in 1951

With great pleasure your Council authorized the award in 1951 of the following Institute Honours:

Awards for Student Essays

The President's Medal and eight cash prizes (each supplemented by a year's prepaid membership in the Institute) were awarded for essays adjudicated in 1951 by the Technical Divisions below mentioned and/or by the Committee on the Medal. The awards were:

Undergraduate Section:

The President's Gold Medal to Jack R. O. Walli, B.Sc., Queen's University '50, for his essay, *The Gieger Counter and its Use in Uranium Prospecting.*

Geology Division: *A Prize of* $25 to Jack R. O. Walli for his essay above mentioned, and *a Prize of* $15 to W. Keith Buck, B.Eng. '50 and M.Sc. '51, McGill University, for his essay, *Geological Mapping, Structural Problems, and the Magnetometer* (see *Transactions* Vol. LIV, 1951, pp. 370-374).

Metallurgy Division: *A Prize of* $25 to John Lund, B.A.Sc., University of British Columbia '51, for his essay, *Slag Control in Basic Open-Hearth Steelmaking,* and a *Prize of* $15 to the Late William J. Losee, B.Sc., Queen's University '50. for his essay. *The Market Outlook and a Survey of the Canadian Copper, Lead, and Zinc Industry.*

Metal Mining Division: *A Prize of* $25 to W. Noel O'Brien, B.Eng., McGill University '51, for his essay, *A Review of Slushing Equipment and Some Applications* (see *Transactions,* Vol. LIV, 1951, pp. 466-474), and *a Prize of* $15 to Amil Dubnie, B.A.Sc., University of British Columbia '51, for his essay, *A Review of Current Operating Practice at the Granby Mine, Copper Mountain, B.C.*

Petroleum and Natural Gas Division: *A Prize of* $25 to Ian C. Grant, B.Sc., McGill University '50, for his essay, *Oil and Gas in the Foothills Area, Alberta.*

Graduate Section:

Geology Division: *A Prize of* $40 to W. Ernest Hale, M.Sc., University of New Brunswick '50, for his essay, *Variation in the Gabbroic Rocks of the St. Stephen Area, Charlotte County, N.B.*

John T. Ryan Trophies

The awards of these Trophies, for outstanding achievement in the prevention of accidents throughout the year 1950, were as follows:

Dominion Trophies:

Trophy for Metalliferous Mines: To the Sigma Mine, Sigma Mines (Quebec), Limited, Bourlamaque, Que.; J. B. Redpath, Assistant General Manager, F. J. O'Connell, General Superintendent.

Trophy for Coal Mines: To the Princess Colliery, Old Sydney Collieries, Limited, Sydney Mines, N.S. (for the second successive year); Frank Heidl, Mine Manager, James Guest, Safety Inspector.

Regional Trophies:

Trophies for Metalliferous Mines in, respectively,

Quebec and East: To the SIGMA MINE, Sigma Mines (Quebec), Limited, Bourlamaque, Que. (*Regional* leader also in 1948); J. B. REDPATH, Assistant General Manager, F. J. O'CONNELL, General Superintendent.

Ontario: To the Ross MINE, Hollinger Cons. Gold Mines, Limited, Holtyre, Ont.; J. J. CATY, Resident Manager; R. D. CAYLOR, Mine Superintendent.

Prairie Provinces and Northwest Territories: To the SHERRIDON MINE, *Sherritt Gordon Mines,* Limited, Sherridon, Man. (winner for eight out of nine successive years); H. H. HALES, Manager, J. R. LAIRD, Safety Engineer.

British Columbia and Yukon Territory: To the SULLIVAN MINE AND MILL, The Cons. M. & S. Company of Canada, Limited, Kimberley, B. C. (*Regional* leader also in 1942); J. R. GIEGERICH, Mine Superintendent, H. R. BANKS, Mill Superintendent, C. F. MYRENE, Mine Safety Engineer, T. S. WILSON, Mill Safety Engineer.

Trophies for Coal Mines in, respectively,

Maritime Provinces: To the PRINCESS COLLIERY, Old Sydney Collieries, Limited, Sydney Mines, N.S. (for the second successive year); FRANK HEIDL, Mine Manager, JAMES GUEST, Safety Inspector.

Prairie Provinces: To the ALEXO MINE, Alexo Coal Company, Limited, Alexo, Alta.; R. TENNANT, Mine Manager, J. CONLIN, Pit Boss.

British Columbia: To the ELK RIVER COLLIERY, The Crow's Nest Pass Coal Company, Limited, Fernie, B.C. (*Regional* leader also in 1948); JAMES LITTLER, Manager, E. CHAPPELL, Safety Supervisor.

Barlow Memorial Medal

For the best paper on Geology published by the Institute in 1950, the *Barlow Memorial Medal* was awarded in duplicate to:

JAMES FENWICK HENDERSON and IRA CHARLES BROWN for their paper, *Structure of the Yellowknife Greenstone Belt, Northwest Territories* (see *Transactions,* Vol. LIII, 1950, pp. 427-434).

The Inco Medal

The Medal of The International Nickel Company of Canada, Limited, was awarded to

C. STEWART PARSONS, *O.B.E.*, for meritorious and practical contributions of outstanding importance to the mining and metallurgical industry of Canada, made during his many years of service on the staff of the Department of Mines, Ottawa, latterly as Director of the Mines Branch.

The Blaylock Medal

The Selwyn G. Blaylock Medal was awarded to

CLARENCE O. SWANSON for distinguished service to Canada in the field of Geology, particularly as regards his method of making quantitative expressions of geological probabilities for use in the control and direction of mining operations and mining exploration.

Honorary Member

In recognition of his distinguished services to Canada in the field of Geology, and particularly toward our knowledge of the Geology of Quebec,

JOHN A. DRESSER, who has been a Member of the Institute since 1902, and who served with distinction as President during the term 1923-24, was, by direction of the voting membership, designated an *Honorary Member.*

Life Members

In recognition of their lengthy membership and/or special services, the following were designated *Life Members*:

GEORGE BOOTH, M.C.I.M. 1923

E. J. CARLYLE, M.C.I.M. 1912, Secretary-Treasurer 1931-49, Executive Director 1947-49. Secretary-Emeritus 1949

KENNETH G. CHIPMAN, M.C.I.M. 1909

HUGH S. SPENCE, M.C.I.M. 1912

OTHER AWARDS

We congratulate the below-mentioned C.I.M. Members, recipients during the year of the awards cited:

The Leonard Medal of The Engineering Institute of Canada, awarded for 1950 to OWEN WILLIAM ELLIS, for the paper, *A Study of Some Alloys of Titanium,* written by Dr. Ellis in collaboration with MESSRS W. E. KUHN and H. V. KINSEY and presented at the 1950 Annual General Meeting C.I.M. (see *Transactions,* Vol. LIII, 1950, pp. 54-67); and

The McCharles Prize of the University of Toronto, awarded for 1950 to WILLIAM STAFFORD KIRKPATRICK, for developing a method to recover sulphur economically from gases of low sulphur-dioxide concentration.

AMENDMENTS TO BY-LAWS

The *Charter and By-laws* of the Institute, published in full in the December BULLETIN, include the amendments to those By-laws affecting (*a*) *Honorary Members* (*By-laws* 2 *and* 11), and (*b*) the annual subscriptions of Members and Associates in all classes (*By-laws* 19 *and* 20), which amendments were adopted during the year by overwhelming, positive votes of the membership.

We deem it important that the increased levies, which became effective on January 1st, 1952, be recorded in this Report. They are as follows:

ANNUAL SUBSCRIPTION RATES

	OLD	NEW
MEMBERS and ASSOCIATES (except those not resident in Canada who are also enrolled in another Institution of the Empire M. & M. Congress	$12.00	$16.00
	10.00	14.00
JUNIOR MEMBERS and JUNIOR ASSOCIATES	10.00	12.00
(except Junior Members for three years after graduation)	3.00	5.00

It is noteworthy that the nominal annual fee of $1 paid by each Affiliated Student-Member has not been increased.

The decision of the membership to contribute voluntarily the additional revenue needed to maintain present services was most gratifying, reflecting, as it does, a widespread, continuing interest in advancing the objectives of the Institute.

Proposed Amendment

During the year, a Special Committee proposed that *By-law* 2*a* be amended so that its provisions respecting those entitled to recognition as *Ex-Officio Honorary Members* shall apply thereafter only to those Ministers of the Crown in Canada concerned, in whole or in part, with affairs connected with the mineral industry. The proposal was submitted for discussion at the recent General Meeting in Ottawa, and will, in June next, be mailed to the membership for decision by letter-ballot.

Suggested C.I.M. Foundation

We next refer you to the important proposal of our colleague, Mr. H. M. Wright, Vice-President for District 6, that consideration be given to the establishment of a *C.I.M. Foundation,* an Endowment Fund having as its objective the expansion of Institute services in all matters pertaining to the welfare of Branches, the membership, and prospective members. This proposal is fully stated in Mr. Wright's letter dated November 26th, 1951, published on page 128 of the Bulletin for March, 1952.

An article accompanying the letter indicates that action toward establishing the suggested foundation was deferred pending receipt of the views of the membership in letters addressed to the Editor of the Bulletin. We urge that members respond to this request for guidance and that Branches discuss and report on the proposal.

Distinguished Service

The President

His colleagues on Council desire to record their deep appreciation of the leadership, services, and time rendered throughout the year by President Hopkins.

It is significant that in the year his ability and accomplishments in the field of petroleum geology and development were recognized by election to the Presidency, he was also called upon to organize the Petroleum Division of the Department of Defence Production, Ottawa. His great talents and executive capacity permitted him to discharge the heavy responsibilities of both offices competently.

His travels on Institute business alone took him to most of the large centres and remote places concerned with mineral production in Canada, and, characteristically, he insisted on visiting such comparatively isolated Branches as the Flin Flon Branch in northern Manitoba, the Yellowknife Branch in the Northwest Territories, and the Mayo Branch in the Yukon. At the conclusion of his year as President, he will have attended three major meetings and addressed the members of at least twenty Branches.

We commend his authoritative address on *Canada's Petroleum Industry,* delivered at the recent General Meeting in Ottawa (see March Bulletin, 1952, pp. 138-147, and *Transactions,* Vol. LV, 1952, pp. 108-117), a noteworthy part of an important symposium on *Canada's Place in the Mineral World.*

The Honorary Treasurer

We congratulate Mr. J. R. Pearson on the *Jubilee* of his active membership and sincerely thank him for his lengthy, devoted service on the Finance Committee of the Institute, for many years as Chairman.

On moving from Montreal to Lennoxville at the conclusion of the 1950-51 Council year, Mr. Pearson requested that he not be re-nominated a member of the Finance Committee and Honorary Treasurer, and it gave your Council much satisfaction to be able to point out that his inability to serve further on the Finance Committee had no bearing on his right to be known as Honorary Treasurer, an honour which, fifteen years earlier, had been conferred upon him in recognition of the numerous years of efficient, voluntary service he had, even that long ago, accorded the membership.

Institute Committees

For special services not cited elsewhere in this report, we sincerely thank the following members and their colleagues on the Institute Committees mentioned:

D. H. McDougall and C. C. Huston, respectively Chairman and Vice-Chairman of the Executive Committee of Council

K. A. Creery, Chairman of the Finance Committee

S. E. Jamieson, Chairman of the Library Committee

K. M. Ralston, Chairman of the Committee on Enrolments

A. V. Corlett and W. R. McClelland, respectively Chairman and Secretary of the Committee on Education for the Mineral Industry

A. O. Dufresne, Chairman of the Committee on the Selwyn G. Blaylock Medal and the Medal of The International Nickel Company of Canada

J. E. Gill, Chairman of the Barlow Memorial Medal Committee

A. D. Campbell and G. R. Yourt, respectively Chairman and Secretary of the John T. Ryan Trophies Committee

John Convey, Chairman of the Committee on the President's Medals for Student Essays

F. V. Seibert, Chairman of the Special Committee on *Ex-Officio Honorary Members*

F. A. McLean, Chairman of the Committee on Publicity

The Technical Divisions

We are pleased to observe that, each successive year, the Divisional plan of conducting the technical affairs of the Institute appears to be working more and more satisfactorily. In this regard it may be said that not only has there been a gradual improvement in the quality and variety of papers presented at meetings and published in the *Transactions* but, equally important, that distribution among a greater number of members of the responsibility for procuring and determining the quality of papers has improved the efficiency of publications' procedure and helped the Divisions to co-ordinate their affairs with those of the Institute.

The Divisions continue to advance their own technologies by special activities such as field excursions, plant visits, and the development of symposia. Another encouraging tendency is an increasing desire on the part of the Divisions to organize joint sessions at major meetings.

In thanking the 1951-52 officers listed below, we particularly mention the services of the Divisional Chairmen, most of whom also acted as Chairmen of the Publications Committees of their Divisions. The most active officers were:

Coal Division: R. D. Livingstone, Chairman; D. G. Burchell, Vice-Chairman; A. Ignatieff, Secretary and Publications representative.

Geology Division: A. W. Jolliffe, Chairman; R. D. Macdonald, Secretary; C. E. Cairnes, Convenor, Sub-Committee on Student Essays; W. G. Robinson, Chairman, Committee on Field Excursions; Sherwin F. Kelly, Chairman, Committee on Geophysics; J. E. Gill, Publications representative.

Industrial Minerals Division: W. G. Cowie, Chairman; T. H. Janes, Secretary, G. M. Hutt, Publications representative.

Metallurgy Division: C. G. McLachlan, Chairman; F. J. Lewis, Secretary; R. J. Traill, Convenor, Sub-Committee on Student Essays; B. S. Crocker, John Convey, and W. E. McFadden, Chairmen, respectively, of Technical Committees on Mineral Dressing, Physical Metallurgy, and Smelting and Chemical Metallurgy; J. U. MacEwan, Publications representative.

Metal Mining Division: H. A. Graves, Chairman; H. I. Marshall, Secretary; G. H. Murray, Convenor, Sub-Committee on Student Essays; L. S. Weldon, Publications representative.

Petroleum and Natural Gas Division: J. B. Webb, Chairman; W. B. Dingle, Vice-Chairman; B. H. Corey, Secretary; J. W. Gregg, Chairman, Publications Committee; J. E. Riddell, Publications representative.

Branches of the Institute

Two new Branches, namely, the Saskatchewan Branch, with headquarters in Regina, and the Nelson Branch in British Columbia, were formed during the year, bringing to *twenty-nine* the number of Branches active at the year end. Subsequently, re-organization of the former Sherbrooke Branch was authorized, and petitions to form at least two additional Branches are pending.

The reports of Branch meetings which appear regularly in the Bulletin continue to reveal the manifold nature of Branch activity, including, besides the presentation and discussion of innumerable technical papers, services to Affiliated Student-Members, courses in prospecting, joint meetings with member groups of other associations, safety promotion, plant and mine visits, and well conducted social functions.

The warmth of the reception given each successive President on his visits to Branches is itself an attestation that the basic strength of the Institute rests in the Branches. Doubtless, also, the contacts thus made do much to keep the Branches vital and strong. However, with growth in the number of active Branches, many of which are widely separated from each other, it is becoming increasingly difficult, if not impossible, for a President to visit each and every Branch during his brief term of office.

Would it not be fair to suggest in this regard that a President would be doing well if he visited *fifty per cent* of the Branches, on the understanding that his successor in office would be expected to complete the circuit? Again, as urged by the 1950-51 Council, the Vice-Presidents and District Councillors are expected to share with the President and Secretary responsibility for stimulating and co-ordinating Branch activity within the six Institute Districts.

INSTITUTE BRANCHES IN 1951

Section or Branch	Chairman	Secretary
British Columbia:		
B.C. Section	H. M. Wright	F. H. Stephens
Britannia Beach Branch	L. L. Allan	A. T. Smith
Nelson Branch	H. E. Doelle	V. M. Killeen
Similkameen Branch	G. L. Mill	E. W. Johnson
Sullivan Branch	H. W. Poole	L. J. Trew
Vancouver Branch	Christopher Riley	F. H. Stephens
Victoria Branch	F. W. Gray	H. C. Hughes
West Kootenay Branch	R. Bainbridge	J. F. Buckingham
Yukon Territory:		
Mayo Branch	N. Gritzuk	R. P. Ehrlich
Alberta:		
Calgary Branch	D. W. M. Ross	A. S. McLatchie
Northern Alberta Branch	E. E. Owen	G. W. Adams
Rocky Mountain Branch	J. J. McIntyre	J. D. B. Brown
Northwest Territories:		
Yellowknife Branch	P. N. Pitcher	J. H. Whyard
Saskatchewan:		
Saskatchewan Branch	R. J. Kilgour	D. J. Sheridan
Manitoba:		
Flin Flon Branch	J. G. Honeyman	Ben Schneider
Winnipeg Branch	J. S. Richards	J. D. Allan
Ontario:		
Cobalt Branch	M. C. Halstead	H. E. Nauss
Kirkland District Branch	Stephen Ogryzlo	George Gray
Lakehead Branch	M. W. Airth	W. H. Cameron
Ottawa Branch	J. M. Harrison	E. R. Rose
Pickle Lake Branch	H. M. Monette	W. D. Ross
Porcupine Branch	W. A. Hutchison	O. S. Atkins
Red Lake Branch	L. H. Van Loon	R. McIntosh
Sudbury Branch	A. E. O'Brien	D. A. Fraser
Toronto Branch	Van H. Smith	G. A. Cavin
Quebec:		
Harricana Branch	H. C. Herz	E. B. Stovel
Montreal Branch	O. C. Wilson	J. E. Carson
Quebec City Branch	E. Larochelle	J. P. Drolet
Thetford Mines Branch	G. W. Smith	C. G. Motherwell
Western Quebec Branch	W. E. Aitchison	M. H. McKelvey

We sincerely thank the many authors and speakers whose papers and addresses contributed so much to the success of meetings, all those who supported Branch activities, and particularly the officers and committees who served during the year. The Chairmen and Secretaries who held office during the greater part of the period are shown in the accompanying tabulation.

Affiliated Student Societies

Including two groups indirectly affiliated through the Mining Society of Nova Scotia, *twenty-two* Affiliated Student Societies were active during the year. Their membership totalled 782, compared with, respectively, 859 in 1950 and 966 in the peak year of 1949. To the date of this report, the 1952 membership of the twenty-one societies still active had reached 594, confirming the downward trend influenced by more normal enrolments at Canadian universities and mining schools.

An increase in the number of Junior Members enrolled during the year indicates that, following graduation, a higher percentage of Affiliated Students are seeking direct membership in the Institute. We attribute this favourable development, in no small part, to the excellent work being accomplished by Mr. G. M. Hutt and his Special Committee on Co-operation with the Student Societies. We also observe with satisfaction that the small increase of $2 in the annual subscription paid by Junior Members for three years after graduation is not adversely affecting current admissions to the Institute of former Affiliated Student-Members.

Only one small group, the *Precambrian Club* at the Lakehead Technical Institute, Port Arthur, became dormant at the commencement of the current year. The twenty-one societies still active, with their current Presidents and Secretaries, are listed in the accompanying table.

Society or Club	University or Mining School	President	Secretary
G. M. Dawson Club	British Columbia	D. L. Russell	R. H. Barnes
Mining & Geological Society	Alberta	L. P. Purcell	J. A. Bordula
Geological Society (*Ore Gangue*)	Saskatchewan	Paul Vogt	C. H. Reisner
Geology Club	Manitoba	R. L. McPherson	R. D. Fetherstonhaugh
Mining & Metallurgical Club	Toronto	W. E. Broughton	R. J. Schwarz
Coleman Geological Club	Toronto	D. P. Stewart	F. F. Langford
Mining & Metallurgical Society	Queen's	F. J. Correy	D. E. Light
Miller Geology Club	Queen's	G. A. McVittie	L. B. Halferdahl
Geology Club	McMaster	S. Roderick	P. G. Sutterlin
Geology & Geography Club (*The Outcrop*)	Western Ontario	W. J. Wilson	R. N. Smith
Mining Geology Club	Provincial Institute of Mining	D. A. McKinley	P. W. Nash
Mining & Metallurgical Society	McGill	T. A. Maki	N. Kipnis
Adams Geological Club	McGill	H. R. Hogan	R. S. Woolverton
Monteregian Geological Society	McGill	E. Parker	K. Defries
Societe de Mines et de Geologie	L'Ecole Polytechnique	A. Thibodeau	P. Brosseau
Institut des Mines et de Metallurgie	Laval	A. Morissette	T. Boisvert
Bailey Geological Society	New Brunswick	W. P. Ryan	M. F. Moore
Dawson Geological Club	Dalhousie	Pearl Goodwin	D. MacQuarrie
Mining Society	N.S. Technical Society	E. A. Holloway	R. D. Mathieson
*Faribault Geological Club	St. Francis Xavier	G. F. Hazel	R. J. Wallace
*Fletcher Geological Club	Acadia	R. A. Crouse	June M. MacLeod

*Affiliated with The Mining Society of Nova Scotia.

The Mining Society of Nova Scotia

As reported at length in the August Bulletin, the *Sixty-Fourth Annual Meeting* of the Mining Society of Nova Scotia, held in Sydney on July 12th to 14th, was, as regards both the quality of the programme and a record attendance, voted one of the most successful conventions in the history of the Society. The meeting was conducted by Mr. J. H. Fraser, retiring President of the Society, whose *Presidential Address* (see *Transactions*, Vol. LIV, 1951, pp. 313-316) was an admirable historical review and appraisal of the progress and achievements of the Dominion Steel and Coal Corporation.

The Society has been doing some very effective work in the field of student welfare and, during the year, sponsored two joint meetings with student affiliates in the Maritimes, which meetings were held at different universities in Nova Scotia. We also note that the Society has voluntarily adjusted its own annual subscription rates, bringing them in line with the new rates adopted by members of the Institute.

The following were elected officers of the Society for the 1951-52 term:

President .. Donald J. MacNeil
First Vice-Pres. .. J. A. Russell
Second Vice-Pres. .. R. Charlick
Sec.-Treas. ..... S. C. Mifflen

Councillors

C. M. Anson — H. Hines
D. G. Burchell — R. M. McColl
T. J. Casey — Angus McDonald
Frank Doxey — W. S. MacDonald
A. E. Flynn — S. G. Naish
Louis Frost — R. P. Nicholson
H. C. M. Gordon — E. W. Paul
K. E. Gustafson — W. S. Wilson

Ex-Officio

J. H. Fraser — J. P. Messervey

Publications

Twelve issues of the Bulletin, Nos. 465-476, were published during the year. These issues, constituting Volume 44 of the Bulletin, contain approximately 926 pages of text and illustrations, exclusive of advertising, compared with 800 pages in 1950.

The said 926 pages of text include 101 technical papers occupying 562 pages, and 364 pages of current news and other editorial matter, including not only the usual Directory of the Institute and Index for the year, but also the *Charter and By-laws* of the Institute.

The audited statements accompanying this report show that, due to the above-mentioned increase in the number of pages and a substantial increase in the cost of printing, revenue from Bulletin operations was somewhat less than in 1950. However, advertising rates were increased, effective from November, 1951, by 17½ per cent, and this adjustment should produce an improvement in operating results for the current year.

Volume LIII, 1950, of the *Transactions*, aggregating 456 pages, was issued in 1951. Demand for the *Jubilee* volume, *Structural Geology of Canadian Ore Deposits*, is continuing.

Your Council appreciates the services rendered during the year by, respectively, Mr. Robert Hendricks, Chairman, and his colleagues on the Bulletin Committee, Mr. L. W. Bladon, Chairman, and his colleagues on the Publications and Technical Programme Committees, the authors of technical papers and special articles, the Editors, and Mr. L. C. Hazelton, Business Manager of the Bulletin.

Membership

Accompanying this report is a statement of membership and a summary of changes during the year.

The total membership, in all categories, was:

At December 31st, 1950 .. 4,468
At December 31st, 1951 .. 4,574

Particularly significant is the fact that steady increases in direct enrolments have more than offset the natural decline (*from* 966 in 1949 *to* 782 in 1951) in the number of Affiliated Students carried in the year end totals. The *Applications Pending* column of the statement indicates that this process is continuing.

Category	Totals as at December 31st., 1950	Changes During 1951					Totals as at December 31st., 1951	Applications Pending, February 15th., 1952
		Enrolments	Removals					
			By Death	By Resignation	By Removal	*By Transfer		
Patrons	2						2	
Honorary Members	7	1					8	
Honorary Members (*Ex-officio*)	14				2		12	
Honorary Members, M.S.N.S.	7	1					8	
Honorary Members (*Ex-officio*), M.S.N.S.	1						1	
Life Members, M.S.N.S.		1					1	
Life Members	97	4	6			1	94	
Members	1,809	118	20	23	29	4	1,851	41
Associates	563	44	8	21	24	4	550	24
Members, M.S.N.S.	172	43	1	12			202	
Corporate Members	66	10					76	2
Corporate Associates	17	3					20	1
Corresponding Members	6						6	
Junior Members	818	(*a*)181	3	14	35	24	(*a*)923	112
Junior Associates	30	9			1		38	6
Totals	3,609	(*a*)415	38	70	91	33	(*a*)3,792	186
Affiliated Students	859	(*c*)782			(*b*)859		(*c*)782	(*d*)594
Totals, All classes	4,468	(*a*)1,197	38	70	950	33	(*a*)4,574	780

*From one category to another. (*a*) Exclusive of 28 enrolments as Junior Members effective January 1st., 1952. (*b*) Student enrolments expire annually. (*c*) Including 38 Student Branch Members, M.S.N.S. (*d*)Number of 1952 Affiliated Students as at February 15th.

Necrology

We record with profound regret the death of the following members:

Alderson, W. P. (1), Montreal,
Beidelman, J. C. (2) , Montreal,
Belec, J. D., Val D'Or, Que.
Bickell, J. P., (1) Port Credit, Ont.
Bolton, L. L. (3), Ottawa, Ont.
Briggs, S. J., Thetford Mines, Que.
Brown, R. C., Montreal, Que.
Cantwell, F. E., Vancouver, B.C.
Chadbourne, H. W., New York,
Coplan, H. P., Ottawa, Ont.
Ennis, R. J. (4), Schumacher, Ont.
Everest, P. B. (5), Bourlamaque, Que.
Ferrier, W. F. (6), Toronto, Ont.
Fletcher, L. K., Los Angeles, Calif.
Gordon, J. P. (2), Renfrew, Ont.
Howells, J. O., Osoyoos, B.C.
Jones, Gomer P. (6), Vancouver,
Jordan, E. H., Toronto, Ont.
Long, Harry V., Managua, Nicaragua
MacAlpine, C.D.H., Toronto, Ont.
MacInnis, Stuart, Sydney, N.S.
MacLean, H. J., Buchans, Nfld.
McTeigue, G. A., Montreal, Que.
Marriott, H. F. (1), Northants, Eng.
Muir, A. K. (7), Toronto, Ont.
Murphy, P. D., Vancouver, B.C.
Neil, Errol K., Montreal, Que.
Norrie-Loewenthal W. G. (8), Vancouver, B.C.
Park, Hugh, Toronto, Ont.
Pollin, F/O W. B., Montreal, Que.
Pullen, E. F., D.S.O. (2), Oakville, Ont.
Puro, R. W., Cobalt, Ont.
Stanley, R. C., New York, N.Y.
Turnbull A. R., Beauharnois, Que.
Weigall, H. S., Paris, France.
Williams, W. E., Vancouver, B.C.
Willis, F. S., Nelson, B.C.
Young, Robert M. (2), Canmore, Alta.

(1) Life Member
(2) Former Councillor
(3) Life Member, former Vice-President and Councillor
(4) President 1946-'47, former Vice-President and Councillor
(5) Chairman, Harricana Branch
(6) Life Member and former Councillor
(7) Councillor and former Vice-President
(8) Former Vice-President

Finance

Appended to, and forming part of, this Report are:

Balance Sheet as at December 31st, 1951; and

Statement of Revenue and Expenditure for the year 1951.

These statements were prepared by Messrs. Haskell, Elderkin and Company, Chartered Accountants, who were appointed Auditors at the last Annual General Meeting.

Your Council certifies that the services of the said firm have been satisfactory.

By Order of the Council,
C. Gerow
Secretary-Treasurer

Headquarters, Montreal,
March 1st, 1952

THE CANADIAN INSTITUTE OF MINING AND METALLURGY

Balance Sheet

31st DECEMBER, 1951

ASSETS

Current			
Cash on Hand and in Bank		$227.32	
Members' Annual Subscriptions Unpaid	$4,163.63		
Less Reserve for Bad and Doubtful Accounts	1,800.00		
		2,363.63	
Sundry Accounts Receivable		218.45	
Investment in Bonds—at Cost		50,378.75	
(Quoted Market Value—$45,095.00)			
Inventory of Stationery and Supplies		744.84	
			$53,932.99
Prepaid Expenses			1,269.80
Advances to Special Volume			2,451.21
Investment in *The Canadian Mining and Metallurgical Bulletin*			1,000.00
Deferred Charge to Revenue:			
Balance of Contribution to Pension Fund for Employees in respect of Past Services			8,693.55
Fixed:			
Furniture and Fixtures	$13,829.44		
Less—Reserve for Depreciation	13,828.44		
		1.00	
Library	9,086.06		
Less—Reserve for Depreciation	9,085.06		
		1.00	
			2.00
Trust Funds—at Cost:			
Life Memberships		$12,960.00	
Barlow Memorial Fund		1,594.57	
Montreal Branch—C.I.M.		3,002.50	
Medal for Bravery Fund		1,474.44	
Retiring Allowances		20,388.11	
			39,419.62
			$106,769.17

LIABILITIES

Current		
Accounts Payable	$3,405.47	
Due to *The Canadian Mining and Metallurgical Bulletin*	1,932.63	
		$5,338.10
Deferred Income and Deposits:		
Prepaid Annual Subscriptions	$2,640.50	
Payment Received in Advance for *Transactions*	670.00	
		3,310.50
Reserve for General Meetings		1,000.00
Reserve for Contingencies		22,000.00
Trust Funds—as *per contra:*		
Life Memberships	$13,025.00	
Barlow Memorial Fund	1,594.57	
Montreal Branch—C.I.M.	3,002.50	
Medal for Bravery Fund	1,474.44	
Retiring Allowances	20,388.11	
		39,484.62
Capital		
Balance as at 31st December 1950	$43,210.37	
Add—Life Memberships Lapsed	675.00	
	43,885.37	
Less—Net Loss for the Year ended 31st December 1951	8,249.42	
		35,635.95
		$106,769.17

Submitted with our Report of this date,

Haskell, Elderkin & Co., *Chartered Accountant,*

Montreal, 8th February, 1952. *Auditors.*

THE CANADIAN INSTITUTE OF MINING AND METALLURGY

Statement of Revenue and Expenditure

FOR THE YEAR ENDED 31st DECEMBER, 1951

ORDINARY REVENUE			
Annual Subscriptions			$42,775.01
Less—Provision for loss of revenue through cancellations, etc.		$1,305.00	
Rebates to Branches—New Members		230.40	
			1,535.40
			$41,239.61
ORDINARY EXPENDITURE			
Salaries and Pension		$32,127.84	
Rent and Taxes		6,019.61	
Postage and Mailing List		1,253.49	
Printing and Stationery		2,026.56	
Telephone and Telegraph		1,033.35	
General Expense		2,094.89	
Travelling:			
Headquarters	$1,677,17		
Members of Council and Committees	338.27		
		2,015.44	
Annual Meetings—net		Cr. 97.25	
Prizes and Awards		746.16	
Audit Fees		500.00	
Cost of Publications—net		1,778.01	
Depreciation—Library	$351.30		
Depreciation—Furniture and Fixtures	2,264.00		
		2,615.30	
Employees' Pension Fund:			
Past Services	$1,738.71		
Present Services	241.81		
		1,980.52	
			54,093.92
Excess of Ordinary Expenditure over Ordinary Revenue (see below)			Dr. 12,854.31
OTHER REVENUE			
Revenue from Investments		$1,942.86	
Bank Interest—net		223.14	
			2,166.00
			Dr. 10,688.31
APPROPRIATIONS			
Grants to Branches		$400.00	
Grants of Life Memberships		700.00	
Provision for Retiring Allowances		2,000.00	
			3,100.00
			Dr. 13,788.31
Deduct—Charges to *Bulletin:*			
Materials and Service	$6,420.00		
Editorial and Other Services	5,400.00		
		$11,820.00	
Less —Credits to *Bulletin:*			
Re Allowance for Members	7,816.00		
Excess of Expenditure over Revenue of *The Canadian Mining and Metallurgical Bulletin* for the year ended 31st December, 1951	68.11		
		7,884.11	
			3,935.89
			Dr $9,852.42
Deduct—Appropriation for *Bulletin* use, 31st December, 1950		$3,603.00	
Less —Transferred to *Bulletin*		2,000.00	
Balance not used			1,603.00
LOSS FOR THE YEAR			$8,249.42

Canada's Petroleum Industry

Presidential Address

By OLIVER B. HOPKINS *

(Ottawa General Meeting, January, 1952)

(Transactions, Volume LV, 1952, pp. 98-107)

PART I

WE HAVE patterned our lives on the things we take from under the earth's surface. Our civilization is built upon the ore from which we obtain our metals and the fuels which power the world as we know it.

Today I should like to discuss with you a newcomer to the ancient mining industry. This lusty infant of which I speak is just a century old — the oil industry.

While one doesn't commonly think of oil men as delvers in the earth, it was by sinking a shaft that the first oil was obtained commercially in Canada. Men sank a shaft in a 'gum bed' in southwestern Ontario's Lambton county, not far from Petrolia, and distilled from the gum a burning oil. That was just a hundred years ago. The gum was originally liquid petroleum which had seeped to the surface and lost most of its volatile components.

Later, the miners dug down to bed-rock and soon after they were not digging but were drilling wells, as they have been ever since. From the wells they obtained crude oil.

What *is* crude oil? It is not an element like gold, silver or lead, but a complex mixture of compounds. These compounds are made basically from two elements, carbon and hydrogen, and are called hydrocarbons (Figure 1).

The remarkable ability of these two elements to combine in different ways gives crude oil its physical and chemical character. The lighter hydrocarbon compounds in it are gases, at normal temperature and pressure, and these are dissolved in the other, heavier compounds. These latter, in turn, range through light volatile to heavy liquids and even to solids, which are also dissolved in the crude (Figure 2).

Crude oil is millions of years old. It had its beginning in the marine plant and animal life which swarmed in the ancient seas that once covered great sections of the earth. The sea creatures died and their remains were covered with sediment which, as the centuries passed, became rock. Imprisoned in the sediment, the organic matter from that ancient life gradually changed to petroleum.

Much of the petroleum formed in the past seeped out of its native rock, escaped to the surface and was lost. But, fortunately for us, not all of it escaped. Some of it became trapped in porous beds underneath a layer of impervious rock. The oil geologist hunts for these rock traps, in the hope that there may be oil in them.

The most usual kind of traps are anticlines, faults, and stratigraphic traps (Figure 3). To find them, the oil prospector observes outcroppings and, especially in our west where

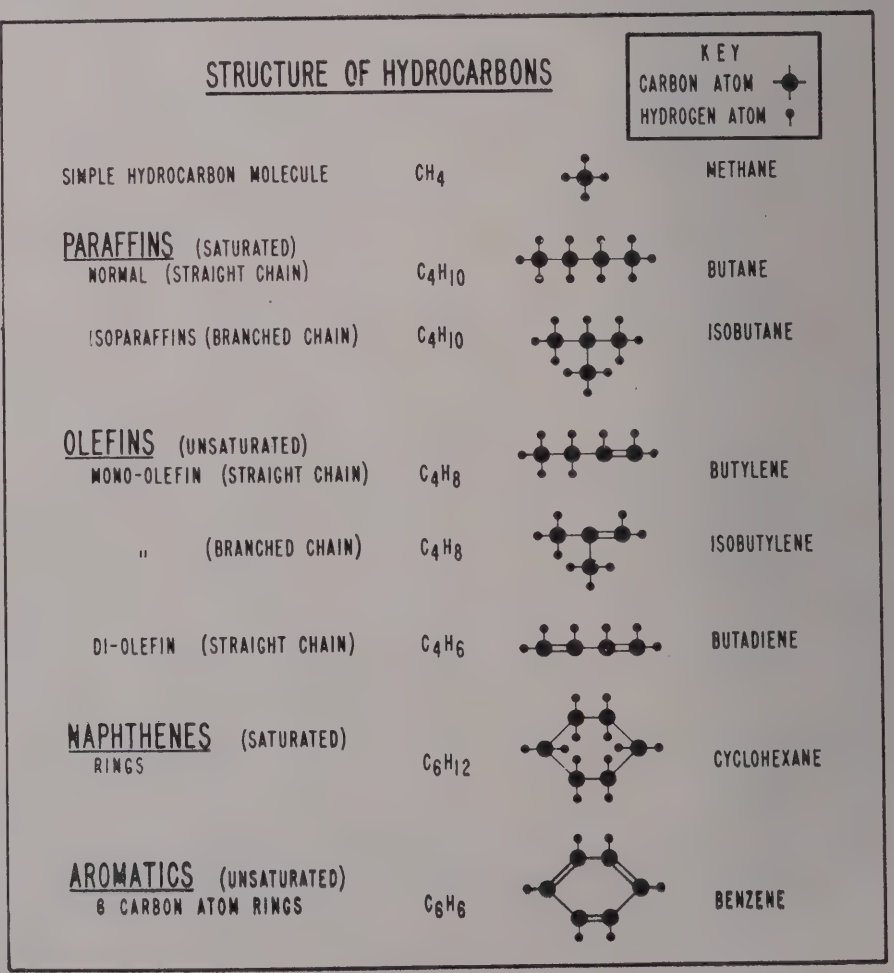

Figure 1

*Vice-President, Imperial Oil Limited.

ORDER OF PRODUCTS		
NUMBER OF CARBON ATOMS	PRIMARY PRODUCTS	PRINCIPAL USES
1 TO 4	HYDROCARBON GASES	FUELS, RAW MATERIALS FOR SYNTHESIS OF CHEMICALS, PLASTICS, RUBBER, ETC.
5 TO 10	NAPHTHAS	AVIATION & MOTOR FUELS, SOLVENTS
10 TO 15	KEROSENE	JET FUELS, INSECTICIDES, LIGHTING, HEATING
15 TO 20	GAS OILS	DIESEL FUEL, DOMESTIC HEATING, RAW MATERIAL FOR CRACKING PROCESSES
20 TO 40	LUBE DISTILLATES	LUBRICANTS, GREASES
	WAXES	COATING, IMPREGNATING, CANDLES, INSULATION
40 TO 100	REDUCED CRUDE	HEAVY FUEL OILS
	PITCH	ASPHALT FOR PAVING, ROOFING, WATERPROOFING, ENAMELS
	COKE	FUEL, ABRASIVES, ELECTRODES

Figure 2

outcroppings are few, resorts to geophysical instruments — seismograph, gravitymeter, magnetometer. Core drilling is also used to determine structure in somewhat the same way the prospector for metals uses exploratory diamond drilling.

When the geologists think they have located a trap, the driller appears with his rig and drills an exploratory or 'wildcat' well down to the prospective formation. And if conditions are right — that is, if oil had accumulated and had been preserved in the trap — then the drill finds a new oil field.

What comes out of the well is seldom immediately useful to man. The refiner must make it so. His raw material varies widely from field to field. Some crude oils have much natural gas in solution, others have very little. Some fields, such as Lloydminster, on the Alberta-Saskatchewan border, produce a thick, heavy oil which, when refined, gives a high yield of heating oils and asphalt. Other fields produce lighter crude which yields much gasoline. Some crudes are 'sour' — that is, they contain sulphur in various forms which must be removed or altered during refining. Some crudes contain a high percentage of lubricating oil components, some more readily yield a high-octane gasoline. A modern refinery can make just about what it wants from any given crude but it is more economical to choose a crude which will most readily yield the desired products.

The processes of modern refining are highly complex but for simplicity they may be considered under two separate headings:

(1) Separation, out of the crude, by distillation, of the components which are useful as such; and

(2) Altering the molecular structures of its components to obtain the desired products; *i.e.*, by cracking, polymerization, alkylation, and reforming.

It is possible to separate the several compounds in crude by distillation, since each compound has its own boiling point (Figure 4). The gases and light products boil off first, succeeded by heavier and heavier oils until only a residue is left.

It is easy to understand how alcohol, which boils at 78.4 degrees centrigrade, can be separated from water, which boils at 100 degrees. This process is allegedly used to some extent in areas beyond the scope of television to produce a brew of considerable potency. Its devotees claim that the product must be aged for at least ten minutes before being consumed.

The refiner's problem is not ageing, but keeping a constant stream of crude flowing through the various units and emerging as high-quality products, made to rigid specifications.

The first step is to heat the crude in a furnace to about 750 degrees F. From the furnace, the hot oil flows constantly into a fractionating tower. While in the furnace, the liquid-gas-oil mixture is confined in pipes, but in the tower the lighter portions of the crude separate and move up the tower in the vapour form.

Across the tower are horizontal trays, pierced with holes into which are inserted short open-ended steel tubes, and over the tubes are serrated caps. The rising vapours are forced to bubble through the liquid on the trays and some of the vapours condense. The higher the vapours rise in the tower, the cooler they become, and thus they condense in their appropriate trays. Meanwhile, the liquid components flow down the tower from tray to tray, becoming heavier and more viscous as they progress, there being sub-

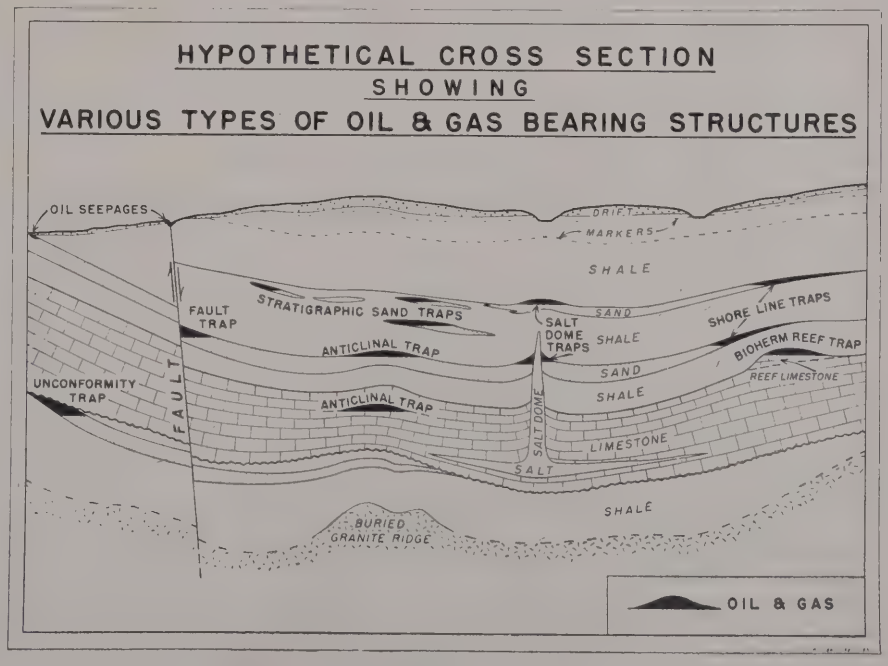

Figure 3

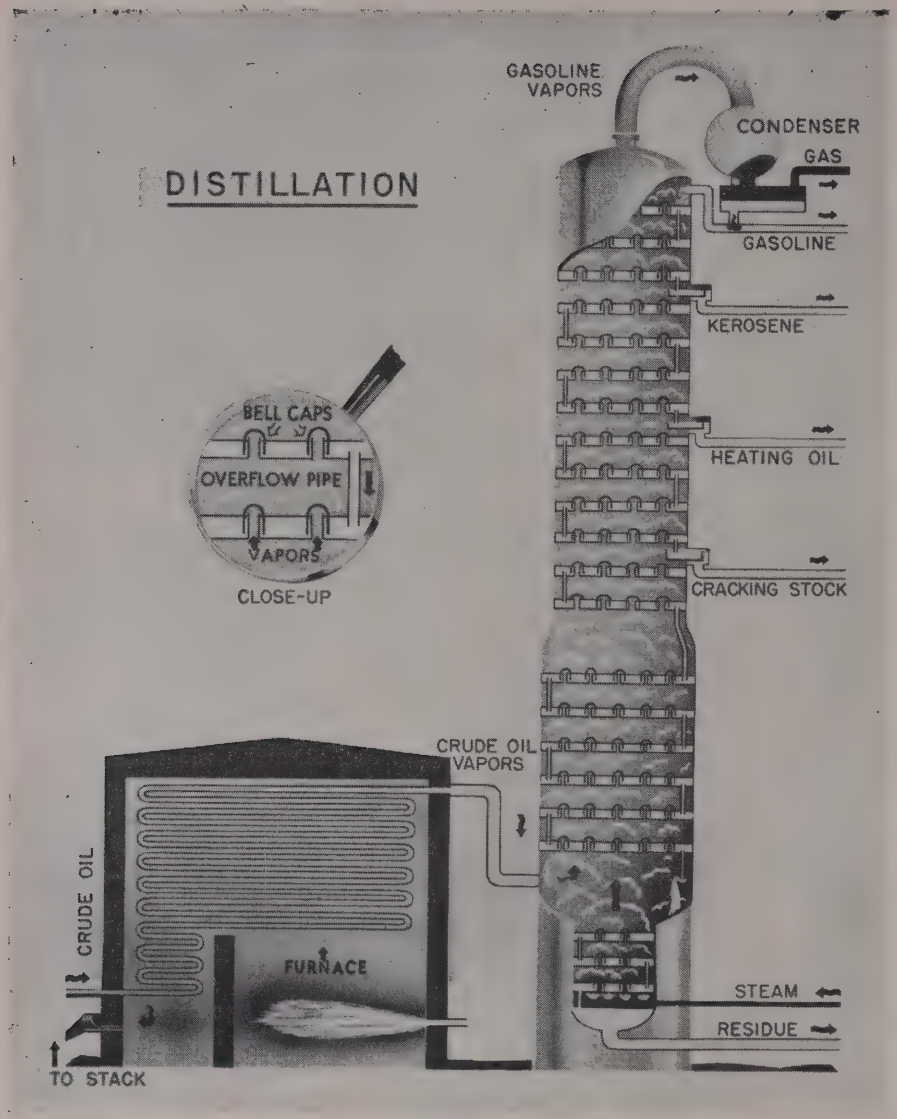

Figure 4

stantial equilibrium at each tray in a well-designed tower. In effect, the tower sorts out the hydrocarbons in the crude oil according to their boiling points.

The fractionation process might be likened to a gravel-screening plant, where crushed stone is graded according to size by a series of screens.

The light naphthas condense near the top, the light fuel oils on lower trays, and so on down the tower, until at the bottom lies the portion of the crude which was not vaporized. This is called reduced crude.

From the side of the tower, condensed kerosene, furnace fuel, Diesel fuel, and so on, may be drawn off. Some of these products are ready for market as soon as they are chemically treated to remove impurities. Others are used for blending and some are subjected to further cracking and/or distillation. Gasoline and the very light products which do not condense in the tower are passed out the top to a condenser, where the gasoline is liquefied. The light gases, which do not condense, are piped to a "light ends" unit, which will be mentioned later.

This means of fractionating crude oil is effective and efficient, but it just does not produce all the gasoline and other products required. The refiner overcomes this problem by doing violence to some of the larger molecules in the crude. Using feed stock from the fractionating towers, he 'cracks' or breaks down the larger molecules into smaller ones and so obtains the required gasoline, plus some heating oils which are also in heavy demand. Here again we can use the analogy of the gravel crusher, which breaks large stones down to the desired size.

One of the latest methods of cracking, called fluid catalytic cracking (Figure 5), involves the use of a solid catalyst, but this catalyst is so finely ground that it will flow when 'fluffed up' with air or gas. At high temperature, the catalyst promotes the cracking reaction of breaking down the heavy oil molecules into lighter molecules that make up gasoline. After it has done its work, the catalyst is separated from the vapours in the large reaction vessel, is regenerated by burning off the carbon, and is used over again.

While there is little to show it on the outside, the interior of a fluid catalytic cracking unit is like a raging infernal hurricane as vast quantities of vaporized oil and powdered catalyst flow through the vessels, mixing and separating at high temperatures. In some large units, the catalyst circulates at the rate of 30 to 40 tons per minute!

Cracking not only produces more gasoline, but also a higher quality product. Today, cracking accounts for about half of the gasoline made by refineries. The cracking process also makes gases or very volatile products known as 'light ends'. These light ends plus those from the crude distillation unit are piped to the light ends unit where, by very precise fractionation, the gases are separated. Butane is blended back with gasoline to give quick-starting and improved anti-knock properties. Propane may be liquefied under pressure and sold as bottled gas.

Certain gases are the basis for the manufacture of synthetic chemicals. At Sarnia, light ends constitute raw material used for making synthetic rubber; ethylene glycol, an anti-freeze; and polystyrene, a plastic.

In the United States, the manufacture of synthetic chemicals from petroleum and natural gas is a huge industry — turning out billions of pounds annually. Here in Canada we have hardly scratched the surface, but several plants are to be built shortly to process petroleum gases into chemicals.

Besides cracking, the refiner has other tricks he can use to get the hydrocarbon molecules he needs. The main three are: polymerization, alkylation, and re-forming.

Polymerization is the reverse of cracking. It involves making a large molecule out of two or more smaller and similar ones. (Figure 6). This is also a catalytic process and is done in a catalytic polymerization or 'cat poly' unit. The feed stock used is selected hydrocarbon gases from the light ends unit and the catalyst is usually tri-potassium phosphate,

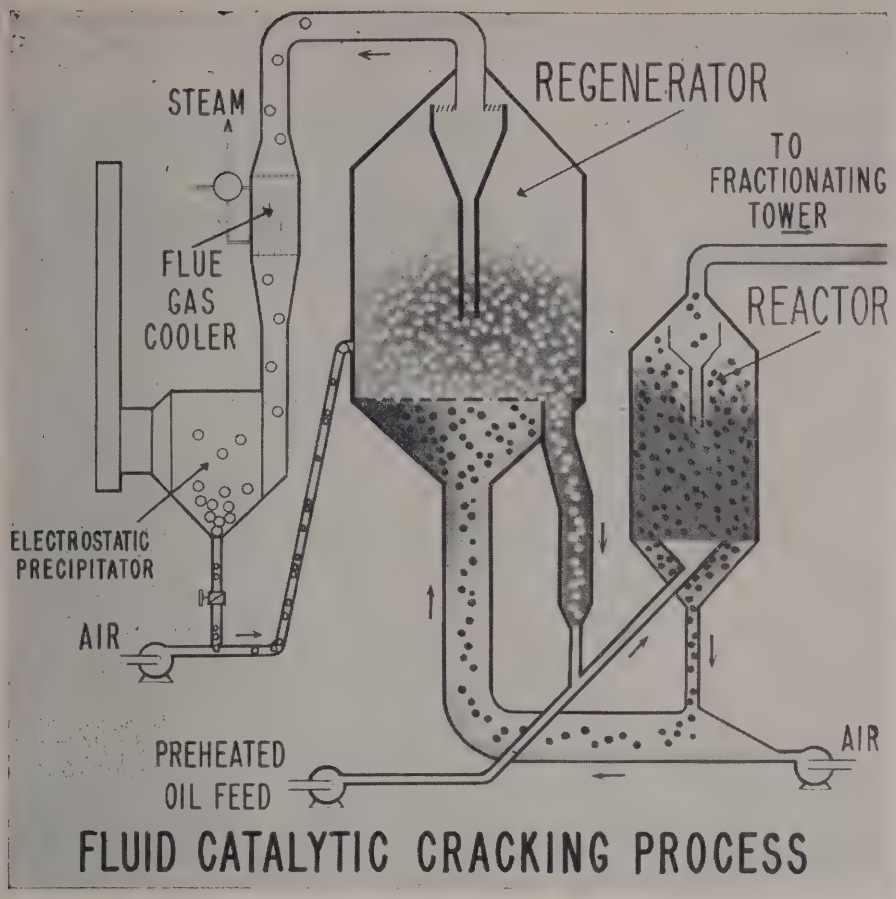

Figure 5

The main product is polymer gasoline — a high-octane fuel which is blended with the other gasolines produced in the refinery.

Alkylation is a word used in the petroleum industry to describe the chemical union of two dissimlar molecules to produce a larger and more useful molecule (Figure 7). That is to say, the refiner takes two hydrocarbons of different families and marries them to produce the kind of offspring he can put to work. As in polymerization, the feed stock for the alkylation unit is generally selected gases from the light ends unit.

The alkylation catalyst is usually sulphuric acid, and the product, a liquid, generally known as alkylate, is particularly suitable for use in internal combustion engines with high compression ratios. Aviation alkylate is the most important component of very high octane aviation gasolines, without which long-range commercial flying would not be possible.

By means of synthesis reactions such as polymerization and alkylation, the refiner upgrades gases, which formerly were of little use except for fuel, into more useful products which increase both the quantity and quality of the gasolines he makes. Synthetic processes are, to a certain extent, an improvement on nature in that we can make compounds that have never existed in nature.

Re-forming, to the refiner, means a process for changing the character of some gasolines. The gasoline which comes from the crude distillation unit — straight-run gasoline — may have a low octane rating. To improve it, the refiner subjects it to high temperatures and pressures, and sometimes in the presence of a catalyst, which re-form the molecular structure to make a gasoline of greatly increased octane rating.

So far, I have not mentioned what happens to the reduced crude which did not vaporize back in the fractionating tower. This is by no means waste, but a highly important material which may be used as feed stock for a thermal cracking plant to produce cracked gasoline and heavy fuel oil; or it may be used to produce other useful products such as heavy gas oil, lubricating oils, waxes, asphalt, and coke.

Very high temperatures would be required to vaporize the lubricating oil fractions under normal pressure, and such temperatures would cause undesirable decomposition of the oil to take place. Therefore, when the refiner distills the reduced crude he employs a high vacuum which permits the heavy oil to vaporize at a lower temperature — thus avoiding cracking or decomposition.

The heavy gas oil may be used as cracking stock or, after treatment, as a very light lubricating oil. The raw lubricating oils may go through several refining steps before they are ready for the consumer. Solvent extraction (Figure 8) is employed to remove components which tend to form sludge and

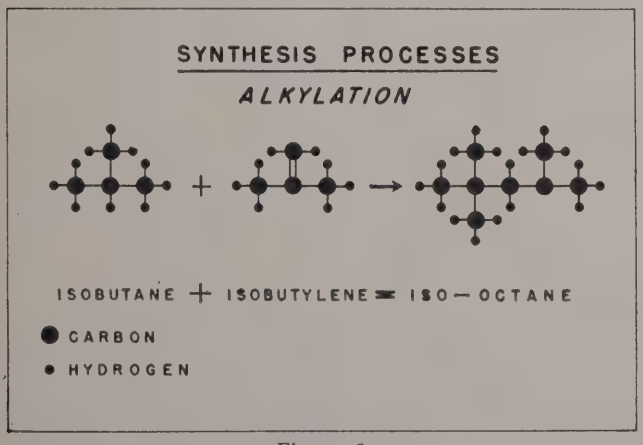

Figure 6

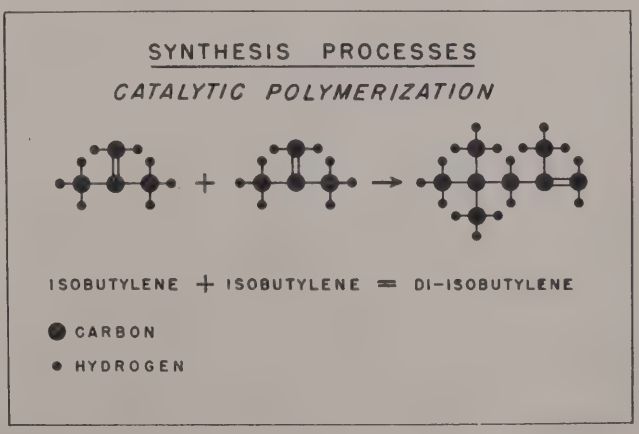

Figure 7

carbon in internal combustion engines. Typical solvents used are phenol, nitro-benzene, furfural, sulphur dioxide, and propane plus cresylic acid. The action of the solvents is to extract selectively the undesirable components and leave behind the more desirable ones.

Many raw lubricating oils contain such a high percentage of wax that they thicken and may even solidify at only moderately low temperatures. Again the refiner makes use of selective solvents to permit removal of the wax. Waxy oil mixed with a solvent is chilled to a low temperature. The oil remains dissolved in the solvent but the wax forms crystals. This mixture then flows to a continuous rotary filter, where the wax is removed. The solvent is recovered for re-use in the process. The wax is further refined into different grades for specific uses.

A third step in preparing lubricating oil stocks is clay treating to improve colour, neutralize mineral acids, and remove other undesirable components. Oil and clay are mixed, heated, and pumped to a continuous rotary filter, where the clay, bearing the impurities, is separated.

Lubricating oils are produced in an amazing variety to meet the many specialized needs of industry and range from oils for delicate instruments to oils for the great bearings of steel mills. These oils are also used in grease making, for a grease is nothing more than a mixture of oil, which does the lubrication, and soap, which holds the oil in place while it does its work. The type of oil determines the lubricating qualities of the grease while the kind and amount of soap determines the ability of the grease to remain in place. Lime, soda, and aluminum soaps are used. Greases, too, have a wide range from clear, almost fluid products to thick, sticky black compounds.

Asphalt may be made hard or soft, depending on requirements. The bulk of the asphalt produced is used for highway or airport paving. When it is used in roofing, the asphalt applied to a flat roof may have a lower softening point than that to be used for a steep roof or for siding, where it might run. Mastic tile, black enamel paint, and battery boxes are some other products in which asphalt is used.

While most people are familiar with gasoline, heating oils, lubricants, and waxes, few outside the oil industry realize what a wide range of products a refinery makes. Imperial refineries, for example, make about 760 different products, and the list keeps lengthening as modern technology makes new demands for fuels, lubricants, and other special products.

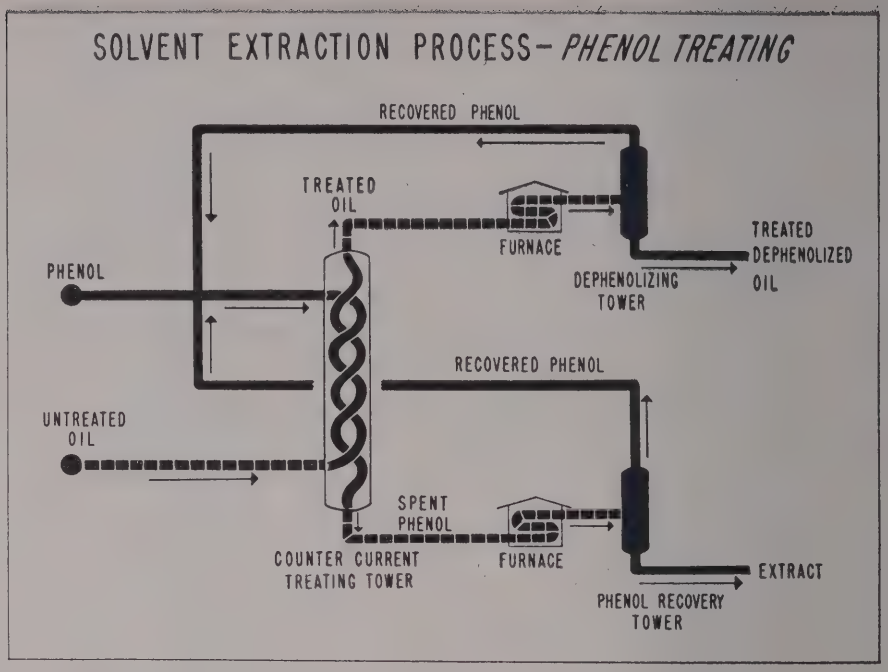

Figure 8

Industrial oils cover a wide range and include oils for metal cutting and grinding, for mining equipment, electrical insulating, coal treating, leather processing, cosmetics, textile manufacturing, and medicine.

Process oils are made for manufacturing cordage, ink, cable dressings and rubber plasticisers, drugs, leather and buffing compounds, insecticides, and floor finishes.

Oil refineries make solvents for rubber cements, paints, lacquers, floor polishes, and also for degreasing hides and for dry cleaning.

The waxes made by a refinery are used for candles, for making paper containers waterproof, and for floor, car, and shoe polishes. Wax also goes into matches, crayons, pencils, carbon-paper, electrical insulation, transcription records, radio parts, and batteries, and is used in photo engraving and in making special printing inks to make them adhere to some surfaces.

Kerosene, the industry's original major product, is still used in oil lamps for illumination and it is also a main ingredient of the fuel used in jet-propelled aircraft. Special illuminating oils are produced for railway and lighthouse lamps, where a clear, bright flame is essential.

Petroleum coke is almost pure carbon and is useful as filling for dry batteries, brushes for electric motors, for arc welding rods, abrasives, and has a place in microphones and in metallurgy for hard-facing steel.

Besides this great variety of products, the refinery also supplies what is called the petrochemical industry with feed stocks. These, as I mentioned, are processed into synthetic rubber, plastics, chemicals, synthetic soaps, anti-freeze, etc.

Not only has modern refinery practice given rise to a large number of new products, but it has improved greatly the products made. Without these improvements, modern automobile and aircraft engines, for example, could not operate. High-compression engines could not run without high-octane fuel, nor would lubricants stand up under the punishment given them by today's engines.

Using high-octane gasoline, an aircraft can not only take off in a shorter distance than formerly, as its engines develop more power, but it can also climb higher and faster and carry more payload. The gasoline must be highly volatile, yet not so volatile that it will cause 'vapour lock', whether the 'plane be flying near the ground or in zones of greatly reduced air pressure several miles above the earth.

Automotive gasoline is tailored to the season's need. In winter, lighter components are added to give it quick starting properties,

but in summer heavier ones are used to avoid vapour lock. The steady rise in octane numbers means that higher compression engines can be used. A modern high-compression engine using premium gasoline can get about twice as much work out of a gallon as was possible a generation ago.

PART II

So far we have considered how oil is formed and how it is converted into useful products to supply today's changing needs. Now let us take a look at Canada's oil industry and its developments. First we shall consider refining, then transportation, and finally the most important function — production.

The refineries of Canada, numbering about forty, have a rated capacity of 389,000 barrels per day, but the Canadian requirements represent some 435,000 barrels of refined products each day. The difference, of course, must be imported as finished products. These are brought in by tanker or tank car. New refineries under construction, and expansion of others, will raise Canadian capacity by the end of this year (1952) to 435,000 barrels a day.

Let us take a look at the location of Canadian refineries (Figure 9). The largest concentration of refining capacity is at Montreal East, where it amounts to about 141,000 barrels a day.

Figure 9 shows also the sources of crude supply for these refineries. As will be noted, some of the crude — that for east coast refineries — comes from as far away as the Middle East. Occasionally, cargoes from Borneo reach our west coast. South America, the United States 'Gulf coast', the mid-Continent and Illinois fields, and California are, however, our major sources of imports.

Keeping the refineries supplied with crude, and moving the finished products where they are needed, and when they are needed, is the task of transportation experts of the oil companies. There are four methods of moving oil — by pipe line, tanker, tank car, and tank truck.

Figure 10 shows Canada's present and proposed pipe lines. In the east, a 236-mile line brings crude oil from a marine terminal at Portland, Maine, to Montreal. Its present capacity is 127,000 barrels a day which can be substantially increased by the addition of more pumping stations. Its use means that Montreal's refineries can operate during the winter when the St. Lawrence is frozen over, without carrying large amounts of crude oil in storage.

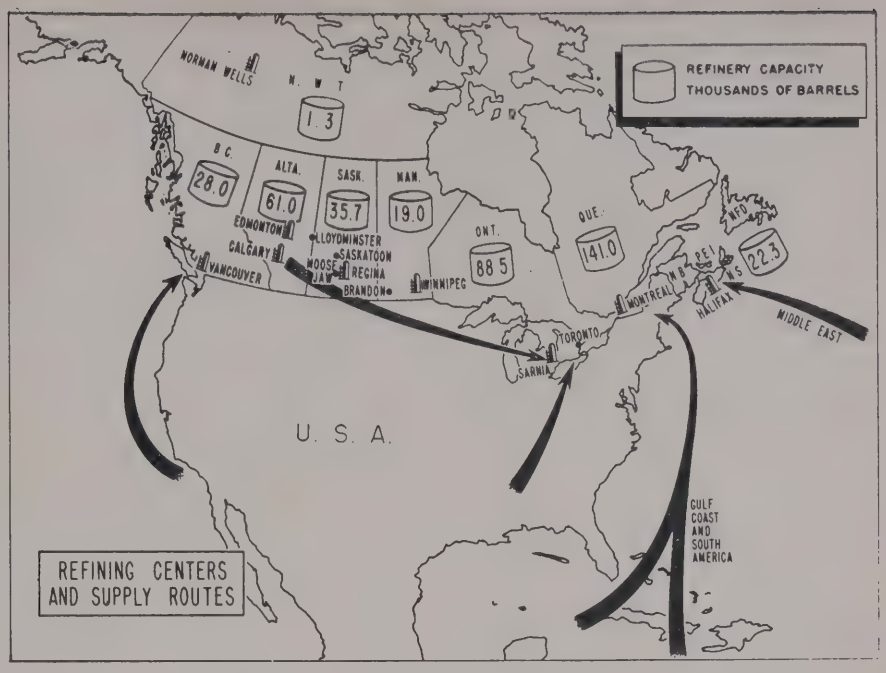

Figure 9

Sarnia's principal source of supply, before Alberta crude reached the east, was a 151-mile line from Cygnet, Ohio, where it ties in with the United States network of pipe lines which reaches as far south as Texas and Louisiana. Capacity of this line is more than 45,000 barrels a day.

Next we have Canada's largest and longest line, the 1,126-mile Interprovincial pipe line from the Redwater field in Alberta to Superior, Wisconsin. This tremendous undertaking was completed in record-breaking time in 1950 and, as soon as navigation opened in the spring of 1951, tankers began loading the Alberta oil for transport to eastern refineries.

This line has an input capacity of 95,000 barrels a day at Edmonton, and of 70,000 barrels beyond Regina. *En route,* the line supplies prairie refineries at Rosetown, Regina, Moose Jaw, Brandon, and Winnipeg. The Winnipeg connection is made through a 75-mile line from Gretna.

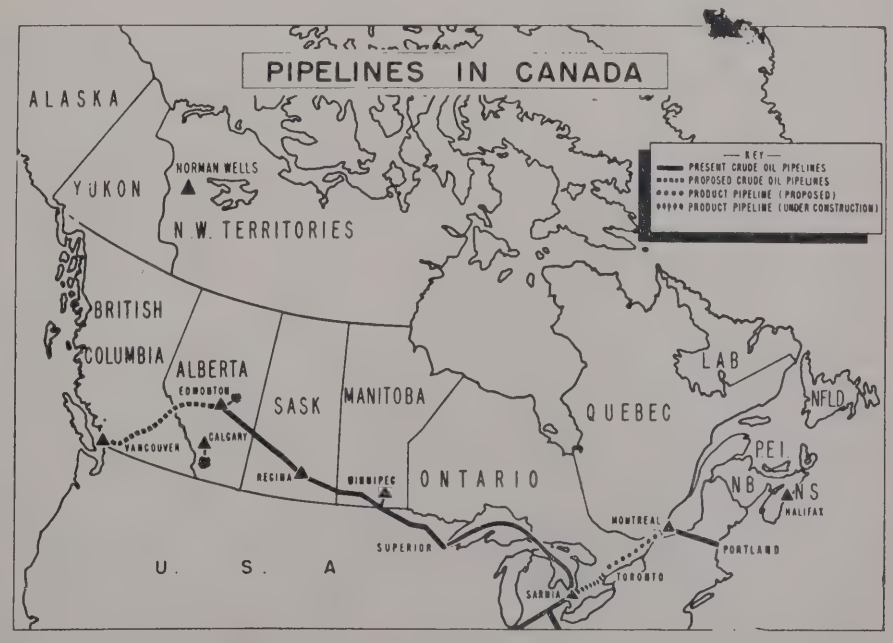

Figure 10

As soon as the original construction was completed, Interprovincial began to increase the capacity of the system by adding five new pumping stations and twelve large storage tanks at Superior, bringing the storage capacity there to more than 4,000,000 barrels.

These new installations will increase the throughput out of Edmonton to 120,000 barrels a day and will permit the yearly movement of about 19 million barrels from Superior to Ontario. This year, an additional pumping station and the laying of a parallel line 100 miles in length will increase yeary anticipated deliveries to Superior to 27 million barrels.

Although this line will operate all year, pumping rates are reduced in winter, when Great Lakes' navigation is closed from December until April. Oil moving through the line in the winter is limited by the storage capacity at Superior and by local deliveries there.

To feed this great quantity of oil into the Interprovincial line, systems of gathering lines are essential. Such systems operate in the Leduc - Woodbend - Golden Spike, Redwater-Simmons, Excelsior, Joseph Lake, Wizard Lake, and Acheson-Stony Plain fields, all in Alberta.

Much older than these lines is the Valley pipe line, built in 1925 to bring Turner Valley oil to Calgary refineries. Capacity of the trunk line is 28,000 barrels daily.

These are the existing lines, but Canada is merely beginning her pipe line development. Various companies have proposals under consideration to lay gas pipe-lines from Alberta to the Pacific coast in the west, and to Montreal in the east, but, as you know, the Alberta government has not yet given its permission for this export. An exception is the export of a specified quantity of gas to Montana, to serve defence industries there.

Several companies have proposed to lay crude oil lines from Alberta to the west coast, to serve both British Columbia and the northwestern United States. One of these companies—the Trans Mountain Oil Pipe Line Company — has recently obtained a permit to build such a line. This Company proposes to start construction in the spring, if pipe is available, and to complete the line in the fall of 1953.

Two lines which will carry refined oil products are being actively planned in eastern Canada. One is a line from Montreal East to Toronto and Hamilton and it is expected to carry about 40,000 barrels daily. The other is from Sarnia to London, Hamilton, and Toronto and will have an initial capacity of 39,000 barrels daily. Construction of the first leg of the latter line — from Sarnia to London — began last autumn and the entire line will be completed this year (1952).

Recent pipe line developments have tended to overshadow the important task which tankers have in supplying Canadians with oil. As I have said, these ships range as far afield as the Middle East and Borneo to bring in crude oil.

On the Great Lakes, and on the east and west coasts, tankers carry products from refineries to marine terminals and move crude to refineries. The movement on the Great Lakes received a tremendous impetus in 1951 when Alberta oil began moving down the Lakes from Superior. The two largest lake tankers in the world were built for this movement, a third is nearing completion, and a fourth is under way.

In regions where pipe lines and tankers are not available, crude and products are moved in tank cars. These cars move wherever rails exist in Canada, delivering the products needed from coast to coast.

Tank trucks may be used to haul crude from small fields, or newly-discovered ones, before rail or pipe line facilities are available. Their greatest use, however, is in delivering oil products to service stations, industries, homes, airports, railways, and transport companies. They are the final link in the chain which starts in the oil well deep underground and runs through the refinery to the consumer.

Naturally, there is a wide difference in cost between the various methods of transporting oil. The modern tanker provides the cheapest transport, but on land the pipe line is supreme, costing about one-quarter to one-third the rail rate.

The effect of modern oil transport is shown in the newly-completed system which brings Alberta oil 1,800 miles to Ontario. To compete in this eastern market, the wellhead price of crude was moderately reduced, thereby producing two results: first, expanding the market for Alberta crude, and second, bringing the benefits of lower prices to western oil consumers. The cost of the transport system reached nearly $100 million for this movement — $90 million for the pipe line and about $4 million apiece for the two new tankers.

Now let us look at Canada's producing industry. Within her borders, Canada has 650,000 square miles of prospective oil territory — that is, the seas, that I mentioned before, covered this large area and deposited a thick section of shales, sandstones, and limestones. These seas, indicated by stippled pattern in Figure 11, extended from the Gulf of Mexico northward through the central United States and western Canada to the Arctic ocean. This area contains the greatest oil development both in the United States and in Canada. Note the extensive development of oil fields in its southern part, largely in Texas and in the adjoining states. Half of the

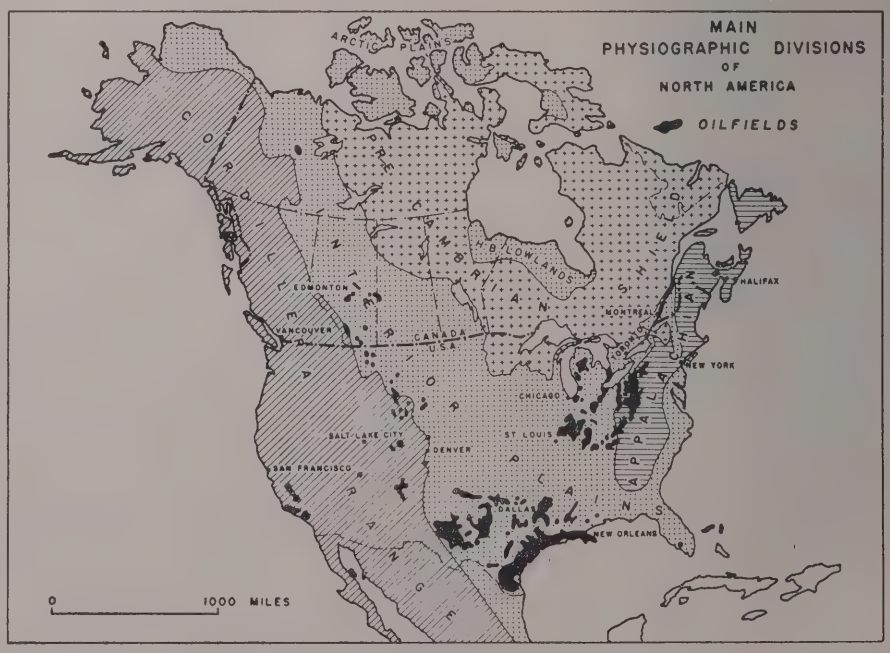

Figure 11

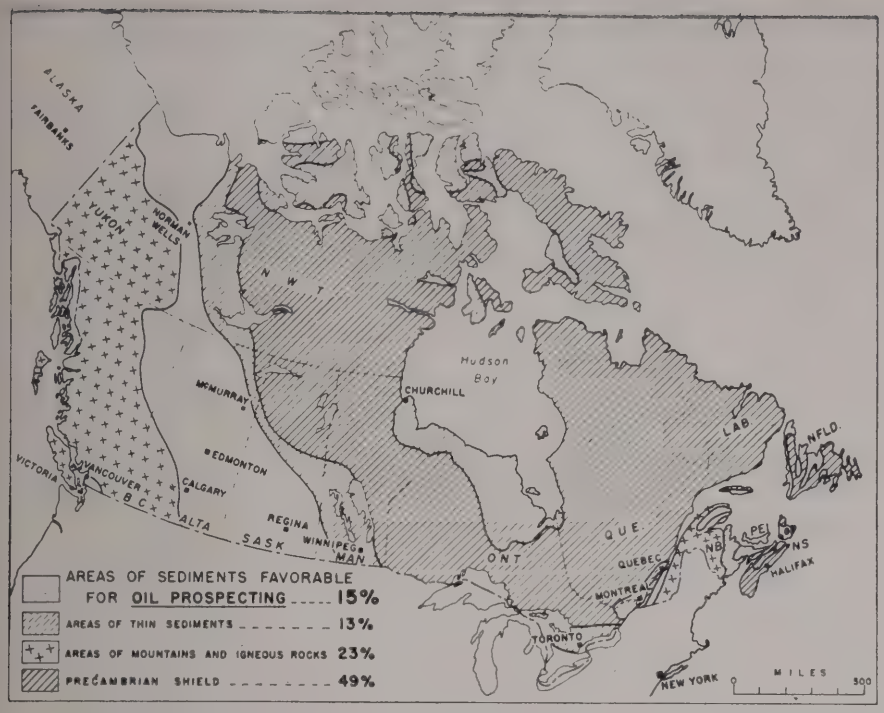

Figure 12

oil production and half of the oil reserves in the United States are located in Texas.

Figure 12 shows in more detail the various areas in Canada that are favourable for oil prospecting, covering about 15 per cent of the total area of Canada. Of the remainder, 49 per cent comprises the Precambrian Shield composed of ancient rocks, which are the source of a large part of Canada's metal production; 23 per cent represents areas of mountains and igneous rocks; and 13 per cent represents areas of sediments that are too thin to offer much promise of oil production.

In the east there are small prospective areas to be found in Newfoundland, Nova Scotia, Prince Edward Island, New Brunswick, and Quebec. The only production found in this area has been near Moncton, in New Brunswick, where some 16,000 barrels of crude were produced in 1950. The southwestern part of Ontario, the cradle of Canada's oil industry, has produced some 30 million barrels and is still producing about one-quarter million barrels per year. In this area, some 20,000 wells have been drilled since the original well was put down in 1858.

It is in western Canada that the largest and most favourable area for oil prospecting is found. It covers a large part of the prairies and extends northward from the southwestern corner of Manitoba to the Arctic ocean (see Figure 13). Farther west there is a small prospective area in southern British Columbia, around Vancouver.

To give some idea of the widespread and extensive activity under way in the west, Figure 13 shows also the distribution of geophysical parties in December, 1951, as compared with their distribution in December, 1946. Some 151 geophysical parties are at present in operation. These are carrying on exploration work over the 168 million acres that have been taken out under lease or exploration permits. Some 223 rotary rigs are in operation over the area shown, of which 107 are engaged in wildcat drilling, and the remainder are on development work in the producing fields.

Last year the industry spent an estimated 215 million dollars in exploration and development work and the search has spread from northeastern British Columbia to southwestern Manitoba. This year the expenditure may reach 250 million dollars, or more than ten times what it was in 1947.

Figure 14 shows the oil fields discovered to date in Alberta and the adjacent parts of Saskatchewan. It shows, in particular, the significant 1951 oil discoveries, which are scattered over a wide area in central Alberta.

To date, some 2,100 exploratory or wildcat wells have been drilled in this great western area. This means that only one well has been drilled for each 195,000 acres of prospective territory. You can see that a vast area remains to be tested. The cost of exploratory drilling is heavy. A wildcat well may cost from $100,000 upward, and the expensive well is just as likely to be a dry hole as the less expensive one.

The exploratory programme calls for the outlay of large sums of adventure capital. Imperial Oil drilled 133 dry holes in the west over a period of 28 years and spent more than 23 million dollars before the discovery well was drilled at Leduc. Leduc, however, was the 'pot of gold' at

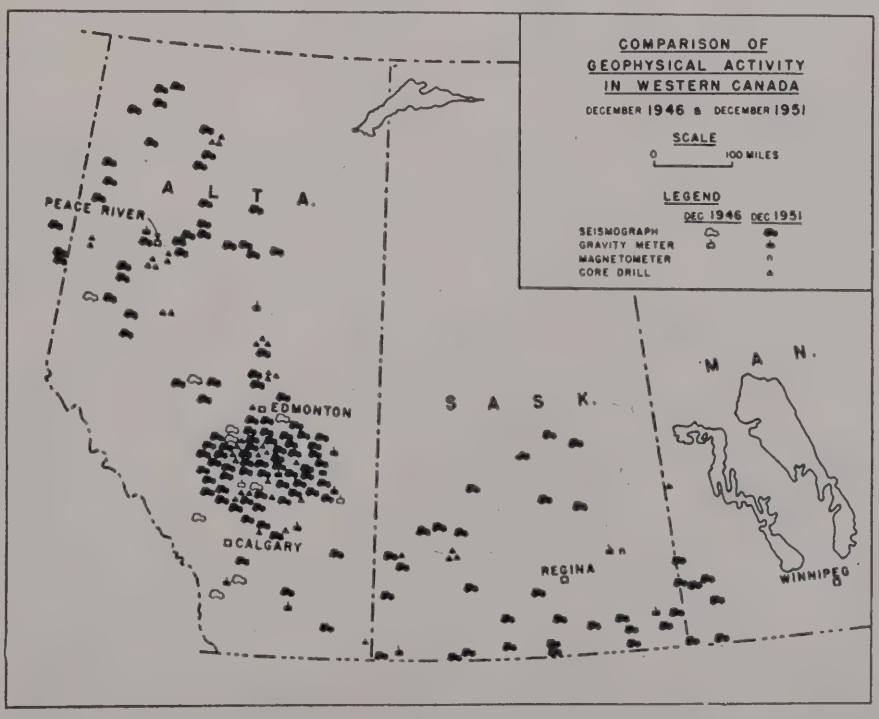

Figure 13

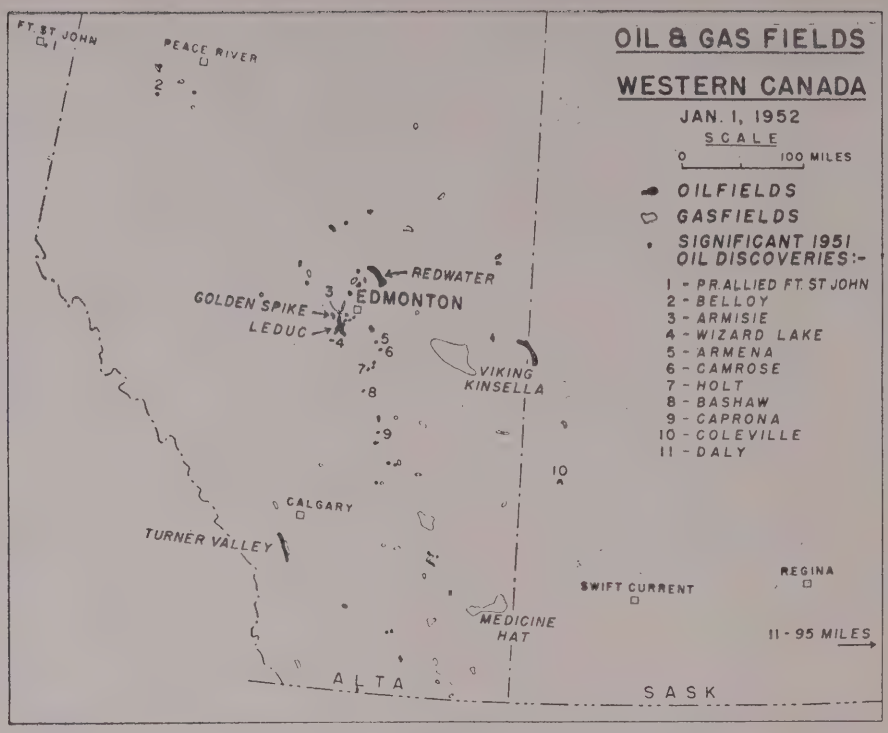

Figure 14

the end of what seemed a most elusory rainbow. It provided proof that major oil fields lay beneath the vast prairie areas. Once the Leduc field was discovered, the exploratory effort increased with such rapidity that it soon doubled and redoubled and doubled again.

Perhaps the extent of the western oil development can best be shown by mentioning a few of the results obtained. Prairie crude oil reserves have increased nearly 30-fold since the time of Leduc, or from 45 million barrels to almost 1½ billion. Production has increased more than six-fold since the discovery of Leduc, from about 20,000 barrels per day to 125,000 barrels. Potential production, or the ability to produce efficiently, has increased 10-fold in the period, from 20,000 to 200,000 barrels per day. Notwithstanding the large increase in consumption, Canada has increased her supply of indigenous production from 8 per cent to 33 per cent of requirements. In 1952 it is expected that some 2,000 wells will be drilled as compared with some 1,500 in 1951. As a result of this drilling, it is anticipated that some 300 million barrels of oil reserves will be found which, when developed, will add 50,000 to 60,000 barrels additional to the potential daily production. By the end of 1953 it is anticipated that the potential production will reach 300,000 barrels per day, which is sufficient to supply 200,000 barrels per day for the prairies and eastern Canada and still leave 100,000 barrels for a western movement.

The year 1951 was a good one for oil discoveries as well as for gas discoveries. During that year, more than 50 gas discoveries were made and the 50 discovery wells had an open flow capacity of some 320 million cubic feet per day.

The tremendous expansion of the producing industry since the discovery of Leduc in 1947 raises the question of how much oil we could expect to find in Canada. It is, of course, impossible to answer this question. Considering the large area available for prospecting and the results that have been obtained to date, it is reasonable to expect that the reserves in the future will be doubled and even quadrupled.

Canadian production must exceed 400,000 barrels a day if the nation is to become self-sufficient in oil (see Figure 15). I doubt whether this country will ever physically supply all the oil she uses, but rather she will become self-sufficient 'on balance'. By that I mean that it would be uneconomic to ship western oil to the maritimes, for example, if we could balance imports there by exports elsewhere. Normally, crude oil is worth more in Vancouver than it is in Halifax, so it would be advantageous to make this exchange.

Already Alberta oil, by replacing imports, has enabled Canada to make important savings in U.S. funds. In 1948, the saving was $9 millions and in 1950 this figure rose to $100 millions. In 1951, with western oil moving to eastern refineries, it is estimated that the saving increased to $150 millions.

Besides the saving in foreign exchange, more tangible benefits have accrued to western Canada. There, as a result of the new oil discoveries, motorists, farmers, home owners, and manufacturers are finding their oil bills lighter. In the prairie

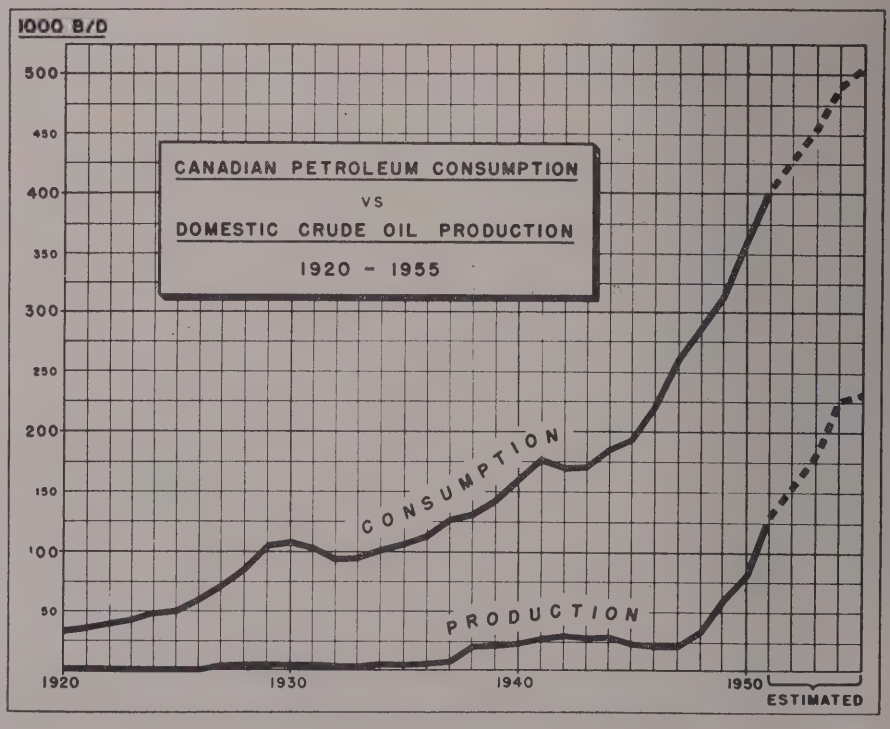

Figure 15

provinces, we estimate this saving to be $55 millions in 1951, or about $22 for every man, woman, and child in Alberta, Saskatchewan, and Manitoba. This represents a saving of five to six cents per gallon to one-fifth of Canada's population.

The oil developments, too, have meant a tremendous upsurge in business activity. Not only have oil firms and their suppliers expanded, but the butcher, baker and candlestick maker have found more dollars flowing into their tills. It is, of course, impossible to estimate the extent of this increased business, but firms from the maritimes to British Columbia have shared in the benefits. And I think it is axiomatic that a healthy economy in the west is beneficial to the nation at large.

Lastly, the new oil discoveries have made the nation more secure in a military sense. During World War I, Canada was almost wholly dependent on imports. During World War II, Turner Valley was the only substantial source of domestic production and frantic efforts were made to develop the Norman Wells field as an additional source of supply. Now we have a large and growing oil supply in our own back yard, so to speak, and we will not have to depend so largely in the future on imports, which in wartime must be tanker-borne over perilous sea routes.

The developing oil picture is so broad and is changing so rapidly that it is impossible to give you more than the highlights of recent activities. It would be a bold man indeed who would at this time predict the future, for even as I speak bits are grinding downward to what might be Canada's largest oil discovery — or another dry hole.

Canada's petroleum and natural gas production now compares favourably in value with that of her other minerals. Last year her oil and gas production had a value of almost $128 million as compared with $159.4 million for gold, $150.6 million for nickel, $149.3 million for copper, and $110 million for coal. The oil and gas industry has become a major industry and has become of national importance since the discovery of Leduc in the spring of 1947.

To bring the Canadian oil and gas industry up to its present state, development has required the expenditure, from 1947 to date, of one billion dollars. Some 575 million has been spent on exploration and development; $115 million on transportation facilities; $60 million on refineries in the west; and $250 million on refineries and other facilities in other areas.

It will require another billion dollars to complete the expansion programme planned for between now and 1955. This programme will go a long way in making Canada self-sufficient in source of crude and in providing the necessary additional transportation facilities and enlarged refining capacity. To provide adequate outlets for the rapidly expanding gas reserves will require some hundreds of millions of dollars in addition. We have been seeing, and will continue to see, a phenomenal growth in the industry, with a corresponding increase in its contribution to the economic life of Canada.

It is customary for the presiding officer of the Institute to speak of developments in his own industry. This is particularly appropriate at this time, as the next annual meeting will be in Edmonton, with emphasis on oil. I have tried to give you a picture of what the baby of the mining industry has been doing, and the importance of his task. He is a vigorous infant, and he needs all his strength for the arduous jobs which lie ahead. He feeds on money and we have seen that he has already consumed almost one billion dollars and that another billion will be required to bring him up to maturity — meaning sufficient crude oil and products to satisfy the national requirement. We firmly believe he will grow to that size.

Progress in Coal Technology

(Series No. 6)

Gasification and Chemical Utilization

*By W. A. LANG**

(*Transactions, Volume LV,* 1952, pp. 108-112)

NOTE.—This is the sixth in the series of coal technology reviews which, commencing in 1948, have been published in the Institute BULLETIN and *Transactions* and have been distributed also, as re-prints, by the Dominion Coal Board.

Topics and authors of the earlier reviews are as follows:

No. 1. — *Mining Practice,* by A. IGNATIEFF, 1948; Nov. BULLETIN, pp. 612-614; Trans., Vol. 51, pp. 262-264.

No. 2. — *Coal Preparation,* by E. SWARTZMAN, 1949; June BULLETIN, pp. 272-275; Trans., Vol. 52, pp. 117-120.

No. 3. — *Mining Practice,* by A. IGNATIEFF, 1950; April BULLETIN, pp. 193-196; Trans., Vol. 53, pp. 133-136.

No. 4. — Combustion and Heat Utilization, by C. E. BALTZER, 1951; Feb. BULLETIN, pp. 89-93; Trans., Vol. 54, pp. 55-59.

No. 5. — *Carbonization,* by W. A. LANG, 1951; July BULLETIN, pp. 472-477; Trans., Vol. 54, pp. 289-294.

INTRODUCTION

CARBONIZATION, gasification, and chemical utilization of coal are interrelated topics, which makes it difficult to review one without indicating its connection with the others. Hence, in *Progress in Coal Technology, Series No. 5: Carbonization,* reference was made to gaseous products recovered in the carbonization of coal, and to a number of chemicals made from the tars, light oils, and ammoniacal liquors which are by-products of coal carbonization.

In this, the sixth paper in the review series, prime consideration will be given to the complete gasification of coal to give a fuel for domestic, industrial, or power purposes and a synthesis gas suitable for the manufacture of gasoline, Diesel oil, and chemicals. The hydrogenation of coal, an alternative source of liquid fuels and of chemicals, will be reviewed, as will be a number of processes for the recovery of chemical products from coal by extraction and chemical treatment.

The reader is referred to Lowry's *Chemistry of Coal Utilization* (1), and Foster and Lund's *Economics of Fuel Gas from Coal* (2) as being two recent comprehensive books on gasification and chemical utilization of coal, and also to *Synthetic Fuel Abstract,* a bi-monthly publication of the United States Department of the Interior (3), which includes up-to-date information on all of the topics included in this review.

GASIFICATION

The complete gasification of coal is carried out usually by the reaction of oxygen and steam with the carbon in the coal to form a gaseous mixture consisting chiefly of carbon monoxide, hydrogen, and carbon dioxide, and, when air is used, of nitrogen. High pressures at relatively low temperatures favour the formation of methane with an equivalent increase in the calorific value of the gas generated.

Coal gasification processes may be divided into three categories: (1) fixed fuel beds, (2) fluidized fuel beds, and (3) entrainment systems. The advantages and disadvantages of each system have been pointed out by King (4) and also by Strimbeck *et al.* (5). Until comparatively recent years, the fixed bed as employed in conventional producer and water-gas plants was the only system used commercially. However, the Germans, in their search for continuous low-cost producers, investigated both the fluidized bed and the gas entrainment technique. In addition, they used coal instead of coke and employed oxygen or oxygen-enriched air for gasification. The principal processes developed by them were: The Winkler, gasifying fines in a fixed fluidized bed; Koppers, gasifying fines in a fixed bed under pressure; Thyssen, Galocsy, and Leuna, gasifying lump fuel and disposing of the ash as a molten slag. The research and development programme in Germany was stimulated by the development in that country of *Fischer Tropsch Synthesis* and *Bergius Hydrogenation* for the production of liquid fuels from coal. Both processes were developed to a commercial scale during World War II and provided much of the gasoline and Diesel fuels used by the army and airforce of that country.

No gasification scheme, however, has met all the requirements for a low-cost process either under European or American conditions and, as a result, there is still a very active research programme on many phases of coal gasification. The following excerpt taken from the paper by King (4) summarizes some of the more important investigations in progress on gasification in England and the United States.

"Classification of Processes

"Group A: Fixed Fuel Bed

" (i) Lurgi-Drawe generator for brown coal. The gasification of reactive brown coal is achieved in oxygen and steam at 20 atm. pressure to give a gas of about 450 B.th.u/cu. ft. calorific value.

"(ii) Texaco Development Corporation. A modified Lurgi generator is operated under pressure but with separate carbonization and gasification zones and including the slagging of ash. Details are not available, but it is stated that any type of coal can be treated, that size of coal is not

*Senior Research Chemist, Research Council of Alberta.

(1) For references, see end of paper.

important, and that the calorific value of the gas is about 450 B.th.u/cu. ft.

"(iii) U.S. Bureau of Mines. A small-scale investigation of the conventional Lurgi generator using bituminous coals.

"(iv) Gas Research Board. Research and semi-investigation of the process.

"The problem is to achieve the production of gas of high calorific value from bituminous coal and, if possible, to raise the thermal efficiency of gas-making by working with an ash-slagging grate.

"Group B: Fluidized Bed

"(i) Winkler generator. The gasification of pulverized reactive brown coal in oxygen and steam at atmospheric pressure to make producer gas and/or synthesis gas.

"(ii) Pittsburgh Consolidation Coal Co. (with Standard Oil Co.). The technical development of the devolatilization and gasification of coal using fluidized beds, so far only at atmospheric pressure.

"(iii) American Gas Association. Semi-scale devolatilization of pulverized bituminous coals to make 'char' or granulated semi-coke for use as a blend constituent in coke ovens.

"(iv) Gas Research Board. Research and investigation of the gasification of pulverized bituminous coal under pressure, so far only with hydrogenation gases, in order to study the mechanism of the reactions.

"(v) Standard Oil Co., New Jersey. Studies of the kinetics of the reactions (so far without publication).

"(vi) Massachusetts Institute of Technology. Studies of reaction kinetics with particular reference to the reaction of C/H_2O and C/CO_2.

"(vii) Imperial Chemical Industries, Ltd. The gasification of fine fuel in suspension (moving burden) by carbonizing in a fluid bed with gas and sufficient air to supply by combustion the heat required to reach 800°C., ejecting the 'char' with air (blowing to incandescence) and steaming in a second fluid bed to make water gas.

"The problem is to apply the boiling-bed technique to the gasification of pulverized bituminous coal with oxygen and steam, and under pressure, to make town gas or methane-rich gas. Operation in stages will be necessary with the rejection of only a low proportion of the carbon of the coal.

"Group C: Entrainment in the Gasifying Medium

"(i) Koppers Process. The gasification of pulverized coal in highly superheated steam and oxygen to make synthesis gas. A modified plant is being erected for the U.S. Bureau of Mines at Louisiana, Missouri.

"(ii) U.S. Bureau of Mines. The modification of the Koppers process to use steam preheated to 3,000°F. to reduce oxygen requirements. The objective is synthesis gas.

"(iii) Institute of Gas Technology, Chicago. Small-scale investigation of the gasification of crushed coal in oxygen and steam using expansion from 50 lb./sq. in. to atmospheric pressure to disrupt the coal, followed by treatment in a 'cyclonizer' to complete the breakdown of the coal and accommodate the gas/solid reactions. The objective is to make synthesis gas, and high steam superheat is intended to be used to reduce oxygen consumption.

"(iv) Fuel Research Station. Research and full-scale experiments at atmospheric pressure on the gasification of pulverized fuel in an induced-vortex with oxygen and steam under pressure, in order to make synthesis gas or hot combustion gases for power production *via* the gas turbine.

"(v) U. S. Bureau of Mines. Semi-scale research on the induced vortex for the production of synthesis gas.

"(vi) Gas Research Board. Research on the fundamentals of the induced-vortex technique and of reaction kinetics (at atmospheric pressure so far, but eventually under pressure to induce the methane-synthesis reactions).

"The problem is to decide whether any of these can be adapted to produce gas of 450 to 500 B.th.u. calorific value. It would appear that the technique holding most promise is that of the induced vortex".

Particular reference is made here to the work by the U.S. Bureau of Mines on the complete gasification of American coals in connection with its synthetic fuels programme. These investigations are being carried out at the Bureau's experimental stations at Bruceton and Pittsburgh, Pa., Morgantown, W.Va., Louisiana, Miss., and Grand Forks, N. Dak. The best sources for information with respect to the design and operating characteristics of the equipment employed, and of the results obtained to date, while gasifying various ranks of American coals under varying conditions of operation, are the *Annual Reports* of the Secretary of the Interior on that Department's synthetic fuels programme, and *Reports of Investigation* of the U.S. Bureau of Mines.

A. R. Powell, Koppers Co. Inc., Pittsburgh, in his introductory remarks at the American Chemical Society Symposium on the Production of Synthesis Gas, New York, 1948, suggested that, from the long-range standpoint at least, the production of low-cost synthesis gas is probably the most difficult and also the most important problem connected with the liquid-fuels programme that must be solved. Later, Newman (6), when summing up the American requirements for a low-cost synthesis gas, suggested that this could best be satisfied by the gasification of low-priced fuels in continuous, internally heated gasifiers when using oxygen or oxygen and steam and with pulverized coal in suspension.

The obvious advantages of the addition of oxygen to air or the substitution of oxygen for air in gas manufacture have long been realized, but economic limitations have prevented the use of oxygen from becoming common practice. However, this barrier seems to have been overcome. Dr. C. C. Wright (7), Chairman of a sub-committee on the use of oxygen in gas manufacture, has made the following statement:

> "The recent advances made in the manufacture of 'cheap' tonnage oxygen stem from the development by Frankl, some 15 years ago, of cold accumulators capable of performing the dual function of air clean-up and cooling more cheaply than could be done by the scrubbing towers and heat exchangers then in use. For the production of high purity (99.5%) oxygen, these cold accumulators or reversing exchangers were not immediately successful because traces of air picked up by the effluent gases limited the oxygen purity to about 98 per cent. For tonnage oxygen production, however, where purity of the order of 90 to 95 per cent is satisfactory, the development was of major significance. Numerous modifications and refinements of the cold accumulator principle have been made in the past few years and these, together with improvements in the over-all cycle and equipment, have made the manufacture of 'cheap' tonnage oxygen an accomplished fact".

It is not surprising, therefore, to find that most of the newer pro-

The construction of two coal-to-oil gasoline synthesis plants has recently been announced. One is being built by M. W. Kellogg and Company for the South African Coal, Oil and Gas Corporation, Ltd. (23), near the town of Coalbrook in the Orange Free State, where Kellogg's 'Synthol process' will be employed, with the operation adjusted to produce maximum gasoline and Diesel oil rather than chemicals. The other plant (24) is planned for the Kalaga Synthetic Oil Corporation, Ltd., at Cuttack, Orissa, India. It will have an initial capacity of 10,000 tons, expandable later to 100,000 tons. Synthesis gas will be made from powdered coal by total gasification.

Chemicals from Coal

Coal is a raw material for a wide variety of chemical substances. These can be produced by carbonization, hydrogenation, extraction, oxidation, and by the synthesis of gases made from coal. At present, carbonization accounts for the largest part of the output but the processes of hydrogenation and of synthesis have greater potential possibilities. The more important chemicals resulting from coal carbonization were reviewed in *Progress in Coal Technology, Series No.* 5, and will not be discussed here.

Chemicals obtained in the high-pressure hydrogenation of coal or coal tars are highly aromatic oils, tar acids, and tar bases. According to Sherwood (25), one ton of bituminous coal will yield on the average 30 pounds of phenol and 70 pounds of cresols plus xylenols. Tar bases are present in smaller quantities and their recovery is economical only in marginal cases. The 1950 Annual Report of the U.S. Department of Interior (9) indicates the effect that the development of a hydrogenation industry may have on the output of chemicals when it states that a total hydrogenation capacity of 30,000 barrels a day, for example, would increase the country's production of benzene and phenol by about 20 per cent and 11 per cent respectively, would provide substantial amounts of toluene and xylene for aviation fuels, and would, at the same time, open up a new source of motor gasoline and liquified petroleum gases.

The nature of the chemical products resulting from the synthesis of carbon monoxide and hydrogen is determined by the catalyst and by operating conditions, notably pressure and temperature. The products are substantially straight-chained paraffins and olefins. Some of the higher molecular weight materials are waxes, while those of intermediate molecular weight are potentially important materials for oxidation to fatty acids and for the synthesis of detergents and plasticizers by the *OXO* process (26, 27).

Another way in which chemicals can be recovered from coal is by solvent extraction. One of the most important materials so recovered is montan wax. The major developments of the montan wax industry have been in Germany where these valuable materials are recovered in large amount from brown lignites by extraction with benzene and/or mixtures of benzene and alcohol. During the war years, shortages of montan wax resulted in intensive investigations in both England and the United States to find coals which would yield sufficient amounts of these materials to make their recovery commercially feasible. The work of the Department of Scientific and Industrial Research of Great Britain (28) is reported in *Fuel Research Technology Paper No.* 52 and that of the U.S. Bureau of Mines (29) in their *Bulletin No.* 482.

A few coals have a high content of fossil resins. These resins are valuable for many industrial purposes, such as the preparation of varnishes, printing inks, waterproofing agents, special synthetic rubbers, and rubber cement. Klepteko (30) has discussed the resins of Utah bituminous coals and outlined methods for their recovery and use.

According to Franke and Kiebler (31), the use of elemental oxygen as the processing medium for the production of useful compounds, principally polycarboxylic aromatic acids, from coal offers an entirely new approach to the age-old problem of transforming the carbon content of coal to more salable chemical materials. The work of the Carnegie Institute of Technology at Pittsburgh is particularly outstanding in this field of research. A process was developed there, and is operating on a pilot-plant scale, for the production of aromatic polycarboxylic acids from the direct oxidation of coal. All the reactants employed are cheap and readily available and every step in the process is apparently applicable to large-scale continuous production. The product, a light coloured powder, is a mixture of aromatic acids, which are water-soluble and strongly acidic. They undergo the usual reactions characteristic of the carboxyl group and have, in addition, the properties peculiar to polycarboxylic acids. Many uses in fields such as general chemicals, plastics, plasticizers, and starting materials for synthesis will undoubtedly be found for this product.

Conclusions

The future of coal gasification and of the chemical utilization of coal appears bright, since many present researches in these fields give promise of wide commercial application. This is particularly true in respect to the production of liquid fuels and chemicals by the gas synthesis and the hydrogenation processes. It seems probable, also, that in the not distant future gas with a heat value comparable to natural gas will be made from coal and at a price competitive in many localities with natural gas. However, to ensure that these developments become realities will require continued support of the many research organizations which are working on the various aspects of these developments.

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cesses are based on the use of oxygen and steam, or oxygen-enriched air. References are noted here to a few of the more recent publications (8, 9, 10, 11, 12, 13, 14) outlining various phases of research investigations on the use of oxygen in coal gasification.

The recent announcement of progress on a gasification project sponsored by Bituminous Coal Research, Battelle Memorial Institute, Diesel Engine Manufacturers' Institute, Pittsburgh Plate Glass Company, Westinghouse Electric Corporation, and General Electric Company is of interest (15). The objective of this project was the development of a producer that would be capable of higher rates of gasification than producers now in use, that would be more compact and more automatic, and that could readily gasify caking and non-caking coals. Dr. E. R. Kaiser, of the staff of Battelle Institute, is the inventor. It is suggested that, if the producer can be used with steam and air, it can be used with steam and oxygen to make a synthesis gas for the production of chemicals. Such an application could make an important contribution to the advancement of the gas producer industry and the science of coal gasification.

Reference was made in *Progress in Coal Technology, Series No.* 3 to the underground gasification of coal.* This topic will therefore not be reviewed here. Suffice to report that a number of important papers on the subject have appeared recently in the technical press. Attention is directed to a paper by Elder and Wilkins (16) which gives an assessment of the Gorgas project, and to one by Masterman (17) which outlines work being done in England on underground gasification. One new development (18) in connection with the underground gasification of coal is worthy of comment. The University of Missouri has carried out experiments on an electrocarbonization scheme. The process takes place in two stages: carbonization effected by passing a high-voltage electric current through electrodes inserted in the seam, followed by gasification of the remaining underground coke bed by air and/or oxygen. The products are a high B.t.u. (550-600 B.t.u./cu. ft.) carbonization gas and a low B.t.u. (150-225 B.t.u./cu. ft.) producer gas.

*See also IGNATIEFF, A., *Underground Gasification of Coal: Review of Progress;* C.I.M., Bull., Nov., 1949, pp. 599-605; Trans., Vol. 52, pp. 265-271.

Synthetic Liquid Fuels from Coal

Hall (19), in a recent paper, states that "world reserves of petroleum oils have always been, and will remain for many years, adequate to meet world demands for liquid fuels, but the unequal geographical distribution of these reserves and the increasing dependence of civilized communities on liquid fuels both in peace and war have stimulated alternative sources of supply. Attention has been directed, therefore, to the production of oil from coal because it is more widely distributed than petroleum".

The two processes already referred to for producing liquid fuels and chemicals from coal have been operated on a commercial scale in Germany and elsewhere, but in neither case have the economics of the process been favourable and the production of oil from coal by them has had to be subsidized. However, according to Schroeder and Fieldner (20), the combined efforts of those working on synthetic fuel processes have yielded major improvements and it now seems reasonable to believe that this is indicative of an industrial development of far-reaching implications. They state that, up to the present, changes in the coal hydrogenation process have been mainly improvements and modifications in the original cycle rather than new process developments. One exception to this generalization is found in the recent development in the U.S. Bureau of Mines laboratories of a process for the dry hydrogenation of coal, or hydrogenation in the absence of a vehicle. This is stated to offer advantages by operation at relatively low pressures and with low plant costs.

The Fischer Tropsch synthesis process (21), as modified in the United States, differs materially from that used in Germany. One of the major changes is that natural gas has been employed as the raw material for the synthesis gas. However, rising well-head costs of natural gas and improvements in processes for the gasification of coal may soon change this picture in favour of coal. Research investigations in connection with the synthesis process centre around the production of the synthesis gas discussed earlier in this paper, and on the development of reaction chambers with high production capacity.

Most of the work done by the U.S. Bureau of Mines on the hydrogenation and gas synthesis processes has been carried out in their Bruceton and Louisiana laboratories. Details of the work, including operational and cost data, are given in the Annual Reports of the Secretary of the Interior, Reports of the U.S. Bureau of Mines, and in papers presented to scientific societies. The information is much too extensive to be reviewed here. However, mention is made of some cost data included in a paper by Schroeder and Fieldner (20) which was read at the Fourth World Power Conference held in London, England, in 1950. The authors stated that an estimate had been made of the initial and the operating costs of a 30,000 bbl./day coal hydrogenation plant using five typical American coals. It was revealed that, from the consumption of 11,830 short tons of Wyoming bituminous coal per day, there would be produced 21,660 bbl. gasoline, 7,100 bbl. liquified petroleum gases, and 1,240 bbl. phenols. The cost of the gasoline, with L.P.G. credited at $0.08/gal. and phenols at $0.10/lb., would be $0.101/gal. The total plant cost would be 243 million dollars, or $8,100/bbl./day.

Some data were also given with respect to the cost of producing liquid fuels by the gas synthesis process but, since the synthesis demonstration plant is essentially new in many respects, these data must be considered only as approximations and may require revision when the pilot plant is actually operated.

In most quarters, the modernized Fischer Tropsch process for the production of liquid fuels from coal is considered as being more economic than the hydrogenation process. This view is not entirely shared by the U.S. Bureau of Mines and it seems likely that, when coal is used as a source of liquid fuels, both processes will be utilized.

Reference should be made to a study undertaken by the U.S. Bureau of Mines Office of Synthetic Liquid Fuels relative to the possibilities of the production of a heavy fuel oil from coal. Two processes were considered: direct hydrogenation, and a combination of hydrogenation with extraction of the coal. The results of this investigation, as given by Hirst *et al.* (22), indicate that, owing to the uncertain status of heavy fuel-oil supplies, it would be economically feasible in some localities to produce synthetic fuel oil if the price of petroleum residuum continues to increase, and that, in other localities, the production of synthetic fuels may become increasingly important.

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Sulphuric Acid and Liquid Sulphur Dioxide Manufactured from Smelter Gases at Copper Cliff, Ontario

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THE world-wide shortage of elemental sulphur, which became sharply apparent in 1950, has focused attention on the commercial exploitation of sulphur-bearing materials such as pyritic ores, sour natural gas, and smelter gases. Although there are no known commercial deposits of elemental sulphur in Canada, there are vast quantities of these sulphur-bearing materials. The International Nickel Company of Canada and Canadian Industries Limited have for many years collaborated in an effort to develop economic processes for the recovery of sulphur† from the smelter gas arising from Inco's metallurgical operation at Copper Cliff, Ontario. While sulphuric acid has been produced at Copper Cliff for over twenty years, recent developments will result in a 60 per cent increase in acid production early in 1952 and, later in the year, in commencement of the operation of the largest single unit ever erected for the manufacture of liquid sulphur dioxide.

The purpose of this paper is to discuss briefly the general factors associated with the recovery of sulphur from the Copper Cliff smelter gases and to describe the sulphuric acid operations and the liquid sulphur dioxide development.

Processes for the manufacture of elemental sulphur, liquid sulphur dioxide, and sulphuric acid from sulphur-dioxide-bearing smelter gases are well known and in some cases highly developed. The products of any such processes must, however, compete in the open market with other sources of sulphur supply. Whether or not they can do so depends upon the cost of production, the availability of raw materials other than the smelter gases, and the location of the smelter with respect to the logical markets and other sources of supply. The cost of production is the only one of these factors that is variable at a given location and efforts at Copper Cliff have therefore been concentrated on devising low-cost processes.

The first success in the Sudbury area came in the middle nineteen-twenties when the Mond Nickel Company, Limited, established the manufacture of sulphuric acid by the contact process at their Coniston smelter (1). This is believed to have been the first contact plant erected to operate on smelter gas. When the Coniston smelter became the property of Inco, in 1929, operation of the acid plant was taken over by C-I-L. Shortly thereafter, a new and larger acid plant was built by C-I-L adjacent to the Copper Cliff smelter and the Coniston plant was closed.

The acid plant at Copper Cliff commenced production in September, 1930, and has operated continuously since that time. It consists of three identical units each having a nominal capacity of 50 tons of 100 per cent sulphuric acid per day. A simplified flow-sheet of the process is shown in Figure 1.

Smelter gas containing, on the average, about 5.5 per cent sulphur dioxide is delivered to the plant through a 5½-foot steel flue at a temperature of about 300°C. The flow of gas is of the order of 18.-000 c.f.m. The operation of the plant may be conveniently described in three steps, namely, gas purification and drying, conversion of sulphur dioxide to sulphur trioxide, and absorption of the sulphur trioxide to produce sulphuric acid.

The gas is first scrubbed with dilute sulphuric acid in two towers operating in series, whereby the gas is cooled and the major portion of the dust carried by it is removed. From the scrubbers, the gas passes through two Cottrell precipitators arranged in series which remove the last traces of dust and also sulphur trioxide mist. Until just recently, platinum was used as the conversion catalyst. Under these circumstances, gas purification was necessary not only to prevent accumulation of dust in the system but to remove platinum catalyst poisons, particularly the halogens and arsenic. The present vanadium catalyst is not as susceptible to poisoning as the platinum but careful purification of the gas remains an important step in the process. It is of interest to note that gas purification was one of the most difficult problems encountered in the original development of acid manufacture from smelter gases at Coniston. Cottrell precipitators were not installed in that plant but the experimental work leading to their use at Copper Cliff was done there. After passing through the Cottrells, the gas is dried by scrubbing with 93 per cent sulphuric acid and is then ready for the conversion step. Immediately after the drying tower there is located a positive displacement-type blower which pulls the gas through the flue, the scrubbing towers, Cottrells, and drying tower and forces it through the remainder of the system.

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†'Sulphur' is used here as a generic name for the industrially important chemicals elemental sulphur, liquid sulphur dioxide, and sulphuric acid.

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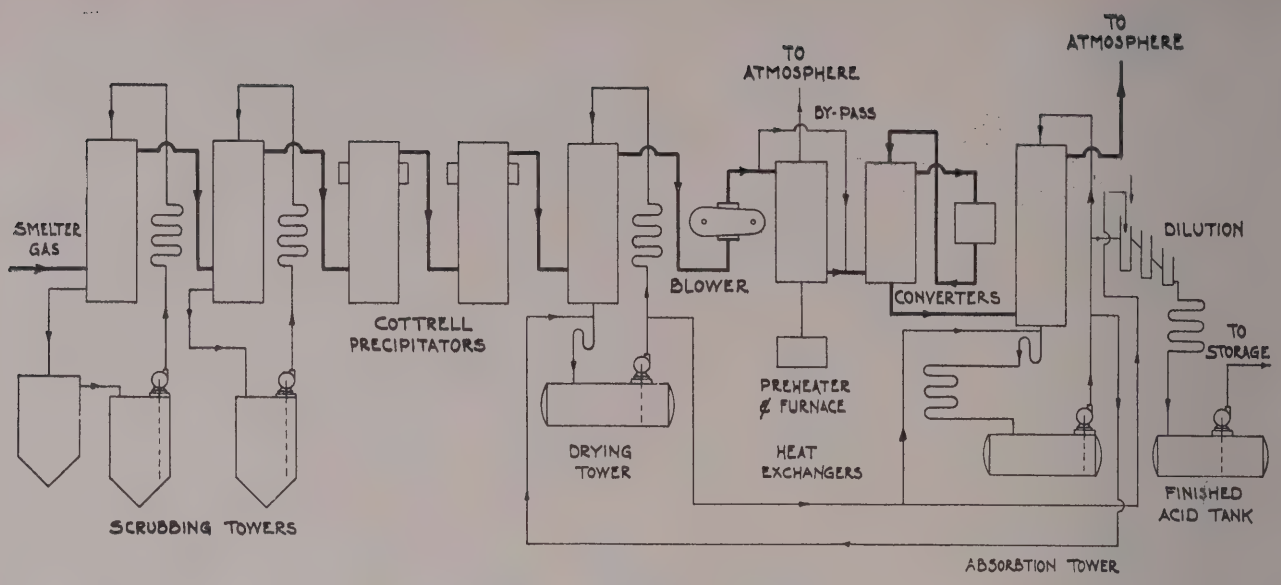

Figure 1.— Sulphuric acid process.

Conversion of the sulphur dioxide to sulphur trioxide takes place according to the simple chemical equation

$$2SO_2 + O_2 \rightleftarrows 2SO_3$$

The reaction is a reversible one and, in proceeding to the right, produces 653 B.t.u.'s per pound of sulphur dioxide consumed. It takes place only at elevated temperatures and in the presence of a catalyst. It has been found that optimum conversion takes place between 350° and 400°C., temperatures in excess of this tending to favour the reverse reaction, as would be expected from the positive heat of reaction.

From the blower, the gas passes through a preheater, heat exchangers, and the converters containing the catalyst. In normal operation, the preheater is not used, since the hot gases emerging from the converter exchange some of their heat content with the incoming gases and by a system of by-passes and controls, the system is maintained at the requisite temperature for maximum conversion. During start-up of the plant or a prolonged period of low gas concentration, the preheater, which is coke-fired, is placed in operation to maintain the temperature of the reacting gas.

The gas from the conversion section of the unit next passes through a tower where it comes in contact with circulating acid containing 99 per cent sulphuric acid which rapidly absorbs the sulphur trioxide in the gas. It has been found that the optimum strength of absorbing acid is 99 per cent because the vapour pressures of sulphur trioxide and water are at a minimum in acid of this concentration. The strength of the acid in the drying tower described earlier is maintained at 93 per cent by the steady addition of 99 per cent acid from the absorption system and, conversely, the strength of the absorbing acid is prevented from rising too high by the addition of the weaker acid from the drying system.

As absorbing acid is produced, it is steadily bled from the system, diluted to 93 per cent acid, and pumped to storage and from thence to tank cars for shipment. Acid is shipped normally as 93 per cent because at this concentration the freezing point is −30°F. while at 99 per cent it is +50°F.

Until recently, sulphuric acid remained the only economic outlet for smelter gases at Copper Cliff. Since 1930, however, other possibilities have been explored almost continually. For instance, C-I-L developed and operated on a pilot-plant scale a unique process for the manufacture of liquid sulphur dioxide from dilute smelter gases. The costs of this process were, however, too high to permit the product to compete with elemental sulphur although the pilot plant was operated for several years to supply the small market for the relatively high priced liquid sulphur dioxide.

In 1946, Inco made a decisive step forward. In that year, the Company informed C-I-L that they were developing, on a pilot-plant scale, a new flash-smelting process which involved the treatment of ore concentrates with oxygen instead of air. This resulted in the production of a gas containing 75 per cent or more sulphur dioxide which, as the starting material, gave promise of economic production of elemental sulphur or liquid sulphur dioxide, since the costly concentration step involved in the use of more dilute gas was automatically eliminated.

Initial studies indicated that the manufacture of liquid sulphur dioxide, rather than elemental sulphur, gave the greatest promise of economic success. Accordingly, C-I-L erected and operated during 1947 and 1948 a pilot plant to provide the technical and economic dat which led to the decision to erect the liquid sulphur dioxide plant now under construction at Copper Cliff. This plant is expected to be in operation toward the end of 1952 and will produce about 90,000 tons of liquid sulphur dioxide per year, equivalent to 45,000 tons of elemental sulphur.

A simplified flow-sheet of the liquid dioxide process is shown in Figure 2. Strong sulphur dioxide gas from Inco's flash-smelting furnace will be delivered under pressure through a 16-inch stainless-steel flue. The gas will first be passed through glass-wool filters to remove the residual dust and then dried by scrubbing with 93 per cent sulphuric acid. Drying of the gas eliminates its corrosive properties and permits of its handling in standard steel equipment. The dry gas will then be compressed in Ingersoll-Rand *XVH Type* two-stage compressors, passed through oil separators, and then cooled with water in tube-and-shell heat exchangers to condense the sulphur dioxide as a liquid. The liquid will be drawn off to storage and the remaining gas will be compressed

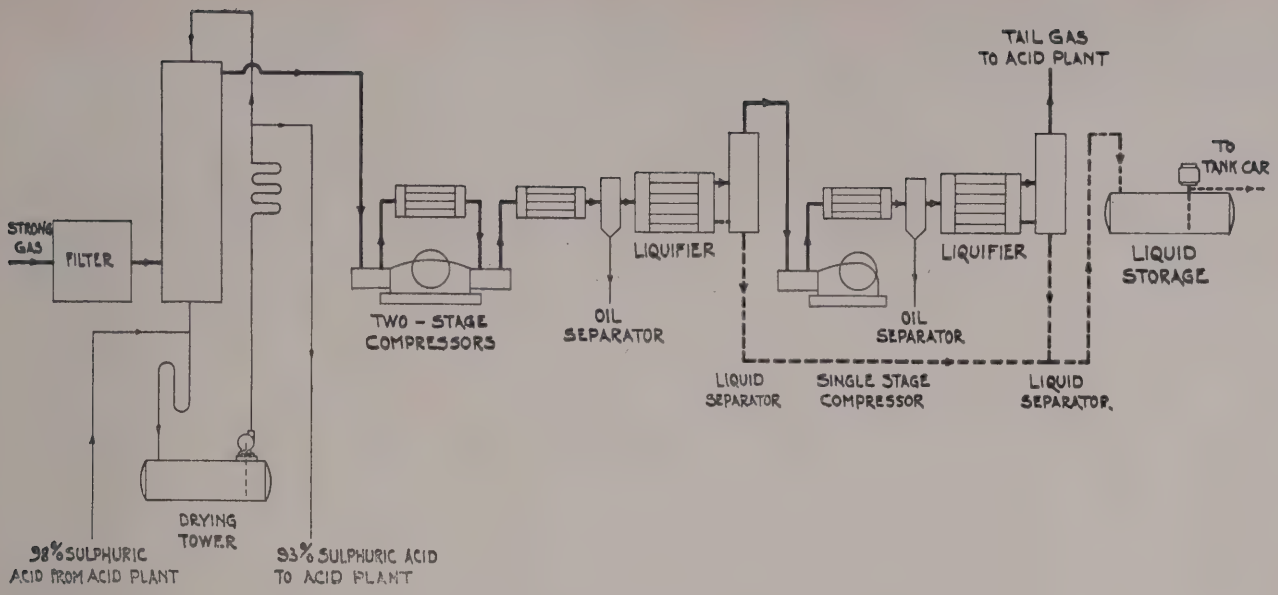

Figure 2.— Liquid sulphur dioxide process.

further, passed through an oil separator and from there to a secondary liquefier, where it will be again cooled with water to liquefy an additional portion of the gas. The unliquefied portion of the sulphur dioxide will be fed back to the acid plant for conversion to sulphuric acid.

It will be noted that there is a close integration between the sulphuric acid plant and the liquid sulphur dioxide plant. The acid plant provides the acid to dry the strong gas and also provides a means of disposing of the tail gas. The economic success of the liquid sulphur dioxide project depends in large measure on this close association with the acid plant.

The liquid sulphur dioxide will be used almost entirely by sulphite pulp mills in Ontario and Quebec. The majority of sulphite mills in Canada now burn elemental sulphur to produce the sulphur dioxide gas required for the preparation of raw cooking acid. On the face of it, liquid sulphur dioxide should be an acceptable alternative to sulphur since sulphur dioxide gas may be produced simply by vaporization of the liquid. Although liquid sulphur dioxide had been used in Europe by sulphite mill operators, it had never been so used on this continent prior to 1947 and the particular mill modifications and operating techniques required were unknown here. In that year, the Abitibi Power and Paper Company undertook to use liquid sulphur dioxide from the Copper Cliff pilot plant to the exclusion of sulphur at their Mission mill at Fort William, Ontario. This trial, which lasted for a period of three months, was entirely successful. It was shown that a sulphite mill may be adapted readily to the use of liquid sulphur dioxide and that there are, in fact, some advantages to be gained (1).

The liquid sulphur dioxide will be shipped from the plant to the sulphite mills in a fleet of 55-ton tank cars. Sufficient fixed storage capacity will be provided both at the liquid sulphur dioxide plant and at the mills to ensure a continuous supply in the event of interruptions in operations or transportation.

This paper would not be complete without mention of the co-operative nature of this liquid sulphur dioxide development, involving as it does three basic industries, namely, metallurgical, chemical, and pulp and paper. Development of the flash-smelting process by Inco provided the initial impetus and the basic raw material, process and economic studies by C-I-L proved the feasibility of production, and the Abitibi Power and Paper Company demonstrated its successful application to sulphite mill operations.

* * *

Contributed Discussion

Mr. Paul Queneau (International Nickel Company of Canada): The following comments cover some of the thoughts that came to mind while listening to Dr. Allgood. It might be mentioned that not only is the C.I.L. sulphur dioxide liquefaction plant the largest such unit ever erected, but I believe its output will equal the total present liquid sulphur dioxide production of the free world. The statement that processes for the recovery of sulphur products from smelter gases must compete in the open market deserves emphasis. It is relatively simple to conceive processes whereby such products can be recovered, but it is vastly more difficult to devise methods which will permit competition with native sulphur at the point of consumption. Over six years will have elapsed since the initiation of the first flash smelting experiment before large-scale liquid sulphur dioxide production will have been achieved. Such a time period for the completion of a major industrial research development is quite normal. Thus interesting investigations which may now be under way will take years to bear fruit. The paper points out that this successful development is the result of co-operation between several basic industries, involving, as it did, Inco, C.I.L., and the Abitibi Power and Paper Company. Credit is of course also due to the gas manufacturing industry represented by the Canadian Liquid Air Company, for its important plant contribution, permitting large-scale production of oxygen at a small fraction of the market price.

(1) Lougheed, E. H., Can. Pulp & Paper Assoc., Tech. Sect. Proc., 1948, pp. 224-228.

Sulphur Recovery from Turner Valley Gas

*By G. A. McGUFFIN**

(*Annual Western Meeting, Calgary, Alta., October,* 1951)

(*Transactions, Volume LV,* 1952, pp. 116-119)

Introduction

WITH the increase in quantity of sour gas reserves in the Province of Alberta, and the wider market for sulphur brought about by mounting world demand and restrictions placed on the principal supply, it was inevitable that this important source of sulphur would be developed. In the light of these facts, it seems timely that a general paper on sulphur recovery from petroleum gases be presented. This, then, is the purpose of the following text. A short discussion of each of the following subjects is presented:

(1) Properties and uses of sulphur
(2) Existing sources
(3) Market research
(4) The recovery process in Turner Valley
(5) Hazards of sulphur production

No pretext of originality of material is made since innumerable reports have been prepared on this and related topics and information contained in these has been used quite liberally.

Properties and Uses of Sulphur

Sulphur is a non-metallic element having an atomic weight of 32.06. The stable crystalline form at atmospheric temperature is referred to as 'rhombic' sulphur. It is bright yellow and has a specific gravity of 2.05, equivalent to 129.2 lb./cu. ft. Above 204°F. sulphur has another stable crystalline form, referred to as 'monoclinic'. This is reddish-brown and is slightly less dense than the rhombic variety (122.4 lb./cu. ft.). Sulphur in bulk form weighs 84 to 90 lb. per cu. ft. Sulphur melts at approximately 240°F. to a straw coloured liquid which is quite viscous. The viscosity decreases with increasing temperature until, at 313°F., it is a minimum. Between 275° and 315°F. the liquid can be handled easily with pumps. Above 315°F., viscosity increases rapidly.

Table I.— Properties of Sulphur

Atomic Weight	32.06
Melting Point	Rhombic, 235°F. Monoclinic, 246°F.
Boiling Point	832.78°F.
Crystallization	Rhombic, stable below 203.9°F. Monoclinic, stable above 203.9°F.
Viscosity	253°F. ... 10.9 centipoises 276° " 8.7 " 301° " 8.1 " 313° " 7.2 " 318.6° " 9.5 " 320° " 22.8 " 320.4° " 77.3 " 329° " 500.0 "
Weight	Rhombic, 129.2 lb./cu.ft. Monoclinic, 122.4 lb./cu.ft. Liquid, 113 lb./cu.ft. at 240°F. Bulk, 84 to 90 lb./cu.ft.

Uses of Sulphur

The uses of sulphur are many. In the United States, approximately 75 per cent of all sulphur is used in the manufacture of sulphuric acid, 10 per cent in the pulp and paper industry, and the remaining 15 per cent in the manufacture of fertilizers, insecticides, carbon bisulphide, and rubber compounds, probably in that order. It should be noted that approximately one-third of the sulphur used in the United States pulp and paper industry is consumed in the Pacific Northwest. In Canada, the principal user of sulphur is the pulp and paper industry, which consumes approximately 70 per cent of the total; 15 per cent is used to make sulphuric acid, and the balance in the manufacture of fertilizer, insecticide, carbon bisulphide, and rubber compounds, in that order. It is worth while noting that about 90 per cent of the sulphur is converted to sulphur dioxide before industrial use.

Existing Sources

The existing sources of sulphur are native elemental sulphur, iron pyrites, sulphur dioxide from smelter fumes, and hydrogen sulphide from natural gas.

Native sulphur mined by the Frasch process (see Figure 1) from Louisiana and Texas sulphur domes accounts for an estimated 95 per cent of all sulphur produced in North America, and, incidentally, for roughly 82 per cent of the world supply. Despite the inescapable inefficiency of the energy utilized in producing sulphur by the Frasch process, the cost of production is low compared to other sources. This fact, coupled with the advantage of coastal shipping, has until recently almost precluded production from other sources.

Iron pyrites or pyrite (FeS_2) is often associated with sulphides of copper, lead, and zinc and may be

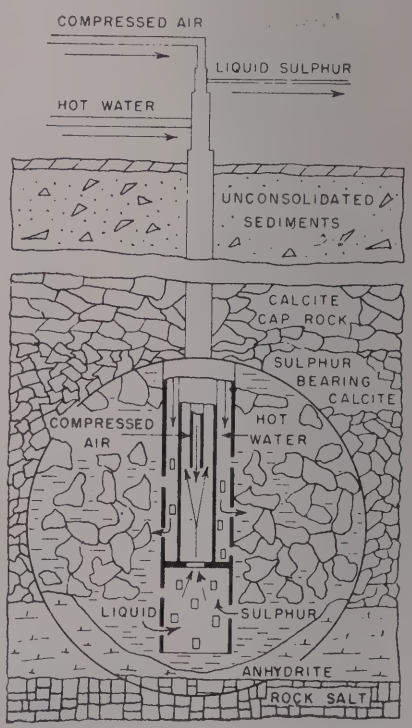

Figure 1.—Frasch process for mining sulphur.

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produced as a by-product in the mining of ores of these metals. As pyrite contains a maximum of 53.4 per cent sulphur, by weight, about 2 tons of the mineral are required for each ton of sulphur or sulphur equivalent produced from it. This practically eliminates pyrite from the elemental sulphur market. Equipment necessary to roast the pyrite for production of SO_2 is expensive, and hence production of sulphuric acid for local markets is the most economical approach. The iron oxides remaining can be used as iron ore.

Sulphur dioxide fumes are most economically converted to sulphuric acid or liquid sulphur dioxide. However, freight rates on liquid sulphur dioxide limit its use to local pulp and paper mills. Sulphuric acid is most economically used locally for fertilizer production.

The production of sulphur from hydrogen sulphide and from the sulphur dioxide produced in the roasting of sulphide ores serves a secondary purpose, that is, it removes the hazard of pollution and contamination of the air. Both H_2S and SO_2 are toxic, H_2S to animal matter and SO_2 more particularly to vegetable matter. Production of elemental sulphur from H_2S is a simple process, and although it requires a fairly substantial capital investment this is not as great as that for the SO_2 recovery processes. Recent developments in the market situation have made production of elemental sulphur from hydrogen sulphide competitive with mined sulphur in Alberta.

Market Situation

Until recently, production of sulphur from domes in Louisiana and Texas has been more than sufficient to meet the world market. The comparative low cost of production plus the advantage of shipping by water ruled out almost all competition. However, in recent years the world demand has been approaching, and in 1950 it exceeded, the supply, the demand in 1950 having been for 6,000,000 tons while production was only about 5,700,000 tons. The North American continent produced approximately 5,000,000 tons, of which only about 300,000 tons was recovered from sour gas. This shortage, plus an allocation placed on sulphur for war industry by the United States Government, has restricted exports from the Gulf Coast and opened the field of sulphur production to other sources.

The chief stumbling block to competitive operations in Alberta has been high freight rates. Now, however, with a premium price for sulphur, this obstacle has been removed and the door has been opened to markets for Alberta sulphur in the Pacific Northwest. Locally, there is a good market for sulphur, which has felt the effects of the restricted imports from the Gulf Coast. The demand is principally for 'crude' sulphur for use in the pulp and paper industry. However, a limited market exists more or less locally for 'commercial' sulphur for use as fertilizer. A recent market survey indicated that, at premium prices, Alberta sulphur could compete also in Manitoba, Western Ontario, and the north central United States.

Development elsewhere of the other sources of sulphur, *i.e.*, pyrite and smelter fumes, is to be anticipated under the circumstances.

The Royalite Recovery Process

Table II gives pertinent data for the Royalite Oil Company's plant which recovers sulphur from Turner Valley natural gas. Following are the principal steps in the recovery process.

H_2S Removal

First step is the removal of the hydrogen sulphide (H_2S) from the gas. It should be noted that in practically all cases this must be done as a matter of course in order to prepare the gas for market. Such being the case, the sulphur now being recovered from the hydrogen sulphide is a by-product.

Table II.— Royalite Sylphur Plant Data

Gas Source: Turner Valley limestone, crude & gas cap

Average H_2S Content: Crude, 1,150 grains/100 cu.ft.
Gas cap, 575 grains/100 cu.ft.
Yearly at plant.
825 grains/100 cu.f't

Gas Scrubbed by Madison:
Monthly average:
Summer (min.), 39,000 M.c.f./day
Winter (max.), 98,000 M.c.f./day
Yearly average:
66,000 M.c.f./day

Average Composition of Sulphur Plant Feed:

Hydrogen sulphide	38.8 vol. %
Carbon dioxide	54.8 "
Hydrocarbons	2.9 "
Water vapour	3.5 "
	100.0 vol. %

Type of Plant:
Foster Wheeler type WTC, single stage.
Erected by Royalite Oil Co.

Design Capacity: 30 long tons per day

Recovery Efficiency:—90% minimum

Quality of Sulphur:—99.5+ % pure

Estimated Completion Date: May, 1952

Approximate Cost: $350,000 installed

The hydrogen sulphide, and also the carbon dioxide, are removed from the gas by scrubbing with an alkaline amine hydrocarbon, usually mono- or di-ethanolamine (see flowsheet, Figure 2). A cool solution of this hydrocarbon will combine quite effectively with H_2S and CO_2, removing them from the gas. This solution, when heated, releases the H_2S and CO_2. It is then cooled and used again. The H_2S and CO_2 gas released from the scrubbing solution is cooled to remove moisture and

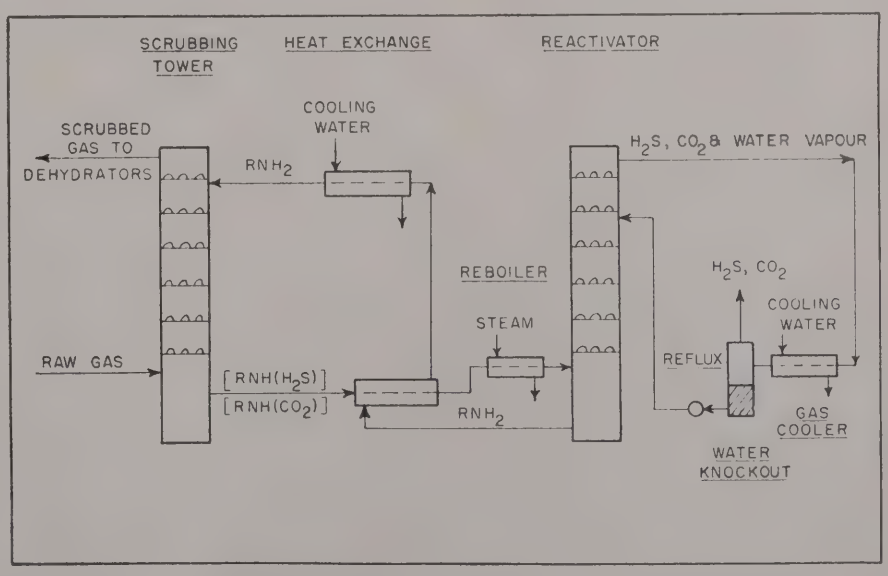

Figure 2.—Typical scrubbing plant for removal of H_2S and CO_2.

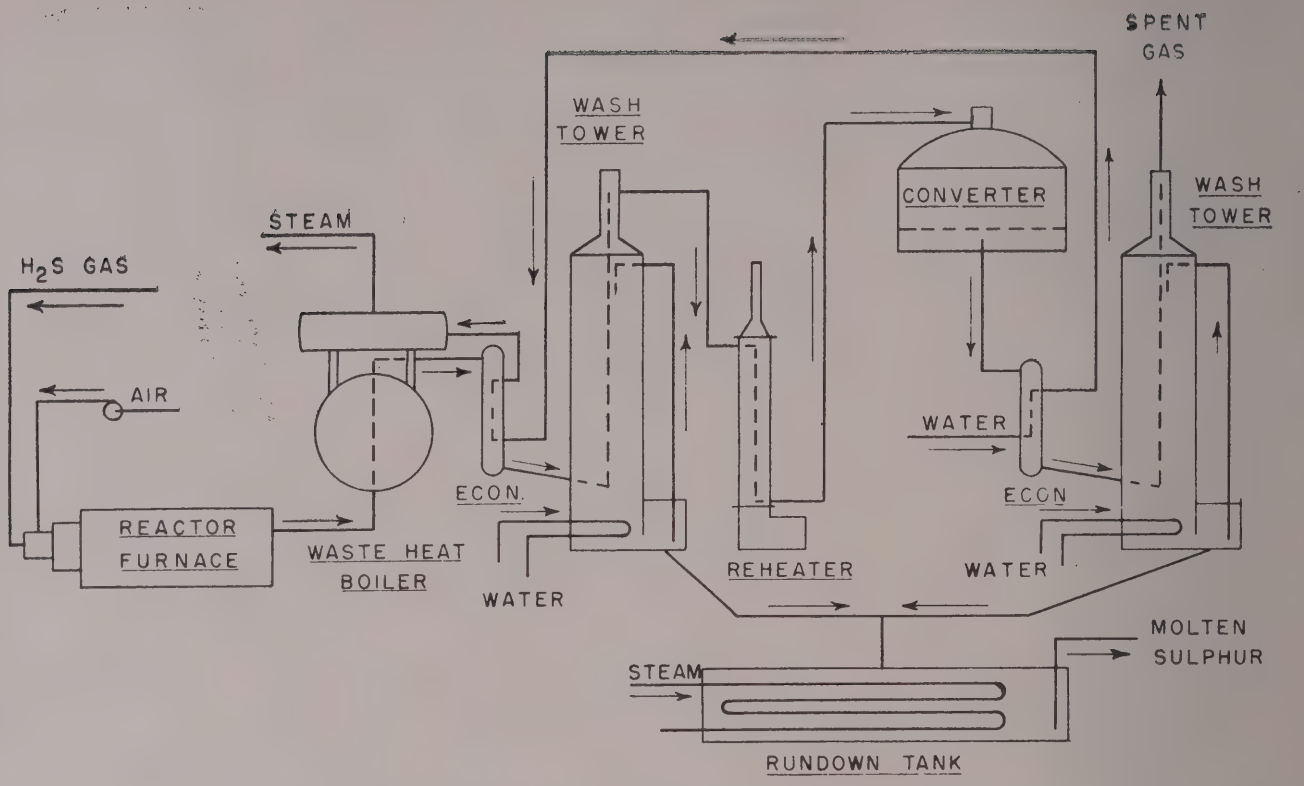

Figure. 3.—Sulphur recovery unit, Foster Wheeler type WTC, single stage.

piped to the sulphur recovery plant.

Sulphur Recovery

The gas ($H_2S + CO_2$) at a temperature of approximately 80°F. is metered and fed into the reactor furnace, where it is burned with a controlled volume of air (see flow-sheet, Figure 3). Approximately one-third of the H_2S is oxidized to SO_2 and any hydrocarbons present are oxidized completely to CO_2 and H_2O. A portion of the SO_2 reacts with some of the remaining H_2S to form elemental sulphur. The equations for these reactions are:

$$H_2S + 3/2\ O_2 \longrightarrow SO_2 + H_2O$$
$$2H_2S + SO_2 \longrightarrow 3S + 2H_2O$$

Approximately 60 per cent of the hydrogen sulphide is converted to sulphur in the reactor furnace and the heat of combustion and heats of reaction are utilized in a waste-heat boiler to produce 225 p.s.i. steam for plant utility requirements. The gases formed, consisting of sulphur in vapour form, unreacted hydrogen sulphide, sulphur dioxide, carbon dioxide, water vapour and other reaction products are further cooled in a boiler feed water economizer to approximately 275°F. At this temperature the gases enter the first sulphur wash tower where all sulphur formed in the reactor furnace is absorbed by a circulated stream of molten sulphur. Off gases from the first wash tower are heated to approximately 500°F. in a natural-gas fired re-heater and passed through a catalytic converter. The catalyst used is bauxite and conversion in this unit increases the recovery to a minimum of 90 per cent. Hot gases from this converter are cooled in a second feed water economizer and passed through a second sulphur wash tower where the additional sulphur formed in the convertor is recovered. The remaining gas, consisting of unreacted hydrogen sulphide, sulphur dioxide, carbon dioxide, water vapour, and other reaction products is vented to the atmosphere.

The sulphur recovered in both wash towers flows into the sulphur run-down tank from where it is pumped to the stock pile and allowed to solidify. Since sulphur solidifies at approximately 240°F., all flow lines are steam traced to keep it in a molten state.

No decision has been reached yet as to handling the sulphur after it is produced. Conventional practice in the past has been to pump the molten sulphur from the run-down tank to the stock pile, where it solidifies. To ship the sulphur, the stock pile may be blasted to break it up, crushed to specification, and loaded by conveyor belt or other means into box cars. Crude sulphur specifications call for lumps not larger than 4-inch. In an alternative method recently developed, the molten sulphur is pumped onto a conveyor-type drying apparatus from the discharge end of which the dry sulphur breaks off in flakes approximately ½-inch thick. This method is very attractive in that it eliminates most of the sulphur dust present in the other method and also because subsequent handling is simplified.

Hazards Accompanying Sulphur Production

Because of the nature of the materials being processed, a short discussion on this subject is warranted.

Toxicity to Workmen

The toxicity of hydrogen sulphide and sulphur dioxide vapours is very high and every precaution against gas leakage must be taken. Past experience in Turner Valley should be valuable in this regard, but new workers should be thoroughly indoctrinated in safety measures.

Fire and Explosion

The chief danger is from sulphur dust ignited by static electricity or other means. Dust must be kept to a minimum, and non-sparking tools, explosion-proof electric motors, etc., should be used. Introduction of the conveyor dryer greatly minimized the dust hazard. Fire

could result from a series of explosions of sulphur dust. Workers should be acquainted with these hazards and instructed as to precautions necessary.

Toxicity of Waste Vapours to Plant Life

Hydrogen sulphide does not present a serious hazard to plant life but sulphur dioxide is destructive to foliage. In the case of the Royalite plant, the waste gases from the process contain very little hydrogen sulphide or sulphur dioxide and, as they will be vented through high stacks, no trouble is anticipated.

Conclusion

The foregoing description has dealt specifically with Royalite's Turner Valley plant. As you are probably aware, one other sulphur recovery unit is now under construction in the Province — conclusive evidence that we are not alone in our market forecast. The probability is that, in the not-too-distant future, at least two additional plants of much larger capacity will be erected. It would appear, then, that the two plants presently under construction herald the development of still another of Alberta's many mineral resources.

The Influence of Roasting Temperature Upon Gold Extraction by Cyanidation from Refractory Gold Ores*

By R. CARTER† and C. S. SAMIS

(*Annual Western Meeting, Vancouver, B.C., November,* 1950)

(*Transactions, Volume LV,* 1952, pp. 120-126)

SUMMARY

The results of this investigation indicate that the gold in the cyanide tailing of refractory gold calcines is occluded in fused particles formed during roasting. This conclusion was reached following microscopic examination of calcines produced at various furnace temperatures with different atmospheres, and a comparison with calcines produced by decomposing samples of completely sulphated concentrate under conditions such that the maximum particle temperature varied between 650°C. and 1,100°C. This comparison showed that the particle temperature during a normal roast may be several hundred degrees higher than the furnace atmosphere, or that the fusion is associated with a low melting (950°C.) eutectic mixture of FeO and FeS.

Because of the unavoidable particle fusion during a furnace roast, and because low particle temperatures are essential for gold extraction by cyanidation, aqueous medium oxidation under high oxygen pressures was studied. Gold recovery from calcines produced by this means was over 98 per cent compared with a maximum of 80 per cent from furnace calcines.

* * *

Introduction

MANY arsenical gold ores are called 'refractory' because much of the gold cannot be satisfactorily extracted by cyaniding the raw ore or concentrate. Even when such ores are roasted to oxidize the iron and arsenic and 'release' the gold, often 10 to 25 per cent of the gold present is lost in the cyanide residues. This problem has engaged the attention of a number of workers during the past twenty years, including Haszard (1), Norwood (2), Leaver and Wolff (3), and Archibald (4, 5). They have sought to develop an explanation of observed phenomena that would account for the presence, in arsenical calcine, of gold that could not be recovered by cyaniding, despite fine-grinding, acid-treatments, alkali-treatments, and other procedures that were applied to the calcine before or during cyanidation.

A review of the ideas and opinions that have been expressed in the literature leads to the conclusion that the reactions involved in roasting merit further investigation if an adequate explanation for the low extraction of gold from arsenical calcines by cyaniding is to be developed. Among the various factors that have been suggested as being responsible are:

(1) That high roasting temperature, as measured by pyrometers at and under the surface of the roasting charge, causes the formation of magnetite which recrystallizes and occludes the gold.

(2) That, by controlling the atmosphere during roasting, solid compounds may be produced in the calcine which have either a beneficial or undesirable influence in the subsequent cyaniding operation.

(3) That, during the course of roasting, some compounds of arsenic, iron, etc., are formed which 'coat' the particles of gold and effectively exclude the cyanide and inhibit solution of the gold.

(4) That the gold is atomically dispersed in the sulphide minerals and remains in solid solution in the calcine and is therefore not accessible to the cyanide solution.

As none of these hypotheses appeared to account for all the observed phenomena, the present investigation was undertaken.

During the progress of the work, the objective was later extended to include the development of a method for evaluating the maximum temperature reached by particles during roasting.

The final objective was to develop a procedure whereby the conditions inhibiting gold recovery could be avoided, making possible the almost complete extraction of gold from a refractory arsenopyrite ore by cyaniding.

Description and Preparation of Materials

The material used throughout the investigation was an arsenopyrite flotation concentrate containing:

Au	1.63 oz./ton
Fe	34.3 per cent
S	33.0 "
As	17.1 "
Sb	0.6 "
Pb	0.8 "
Cu	0.3 "
Zn	3.0 "
Insol	8.3 "

The behaviour of this concentrate when cyanided 'raw' is shown in Table I. Tests *C-1* and *C-2* show that 21.5 per cent of the gold can be extracted by direct cyanidation of the concentrate, and that the extraction is increased to 31.1 per cent when the concentrate is ground for one hour before cyaniding.

It was also found that this arsenical concentrate, when roasted in air at furnace temperature of 450°C. and 750°C., and cyanided, yielded gold extractions of 67 per cent and 16 per cent, respectively, as shown in Table I.

Throughout this report, gold recoveries effected by cyanidation were calculated on the basis of a calculated head. The assay of the

*This work was supported by the Research Fund of the Consolidated Mining and Smelting Company of Canada, Limited, and by a National Research Council Bursary held by the junior author.

†National Research Council Fellowship, Royal School of Mines, London, England.

§Professor of Metallurgy, University of British Columbia.

(1) For references see end of paper.

TABLE I.—GOLD EXTRACTIONS EFFECTED BY CYANIDATION

TEST NO.	TREATMENT	TAILING Au oz/ton	Au EXTRACTION %
C-1..	Raw concentrate as received.....	1.28	21.5
C-2..	Raw concentrate, 1 hour grind in pebble mill before cyaniding.....	1.12	31.1
C-3..	Raw concentrate, roasted 11 hours at max. 450°C....	0.53	67.0
C-4..	Raw concentrate, roasted 3 hours..	1.37	16.0
C-44.	'Distilled' concentrate, unroasted..	1.08	45.0

tailing cannot be used to calculate gold recoveries as the weight of the calcine to the starting material varied widely. The assays of tailing are reported to permit direct comparison on a qualitative basis.

In order to eliminate the arsenic as a major factor, the arsenical concentrate used in the present investigation was treated in the following manner before roasting:

2,000-gram charges of concentrate were heated in a retort, under a nitrogen atmosphere, for one hour at 600°C. The arsenic was effectively removed as vapour and sulphide by this distillation, leaving a porous residue ('distilled' concentrate) having the following analysis:

Au	2.3 oz/ton
As	0.6—0.8 per cent
Fe	50—52 "
S	32—33 "

TABLE II.—GOLD EXTRACTIONS EFFECTED BY CYANIDATION
DISTILLED CONCENTRATE ROASTED IN AIR AT DIFFERENT FURNACE TEMPERATURES

TEST NO.	MAX. TEMP. °C.	TIME AT MAX. TEMP. hr.	CALCINE		RESIDUE	
			S-SULPHIDE %	S-SULPHATE %	Au oz./ton	Au EXTRACTION %
C-53.....	450	3	5.5	4.2	0.48	79.8
C-54.....	500	1.5	4.7	1.3	0.54	76.6
C-47.....	550	0.5	1.2	1.3	0.84	60.9
C-37.....	600	0.5	2.0	1.7	0.98	56.2
C-34.....	750	0.5	0.7	0.2	1.58	16.3

The distillation thus removed approximately 96 per cent of the arsenic and 33 per cent of the sulphur.

This porous product, shown in the accompanying photomicrograph (Figure 1), from which 45 per cent of the gold could be removed by direct cyanidation (Test C-44 in Table I), was used as a base material for all subsequent roasting tests.

EXPERIMENTAL PROCEDURE

Roasting

In all tests, 1,000 to 2,000 gm. of 'distilled' concentrate was roasted in silica trays sealed in a fire-clay muffle as shown in Figure 2. This assembly was placed in an electric muffle furnace in which the temperature was controlled by a thermocouple placed under the silica tray. The furnace atmosphere was controlled by introducing the desired gas through the tube shown at the back of the muffle. The usual procedure followed was to introduce the charge to the muffle, which had been previously heated to 400°C. — the lowest temperature at which roasting would begin — increase the temperature at the rate of about 100°C. per hour to the desired maximum temperature, and then retain the roasting charge at the maximum temperature for a predetermined period.

(*b*) *Cyaniding*

The following standard cyanidation procedure was adopted:

(1) One hundred grams of the calcine was washed for one hour with 500 ml. of water in an open five-litre wide-mouth jar with ten small pebbles and revolving on rolls at 10 r.p.m. The calcine was then washed twice by decantation from two liters or by filtration on a Buchner filter.

(2) The washed calcine was returned to the original jar and cyanided on the above mentioned rolls for 24 hours with 500 ml. of water, 0.5 g. of NaCN, and 1.0 g. of CaO.

EFFECT OF FURNACE TEMPERATURE ON GOLD EXTRACTIONS EFFECTED BY CYANIDATION

To ascertain the effect of temperature on gold extraction, a series of tests was run in which there was free access of air to the charge and in which the total roasting time and other conditions were kept sub-

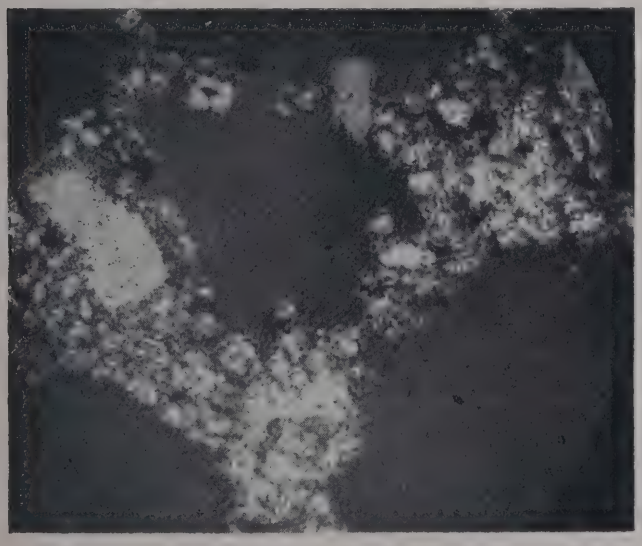

Figure 1.—Distilled concentrate. Shows porous structure of distilled concentrate and dense particle of undistilled arsenopyrite. X375.

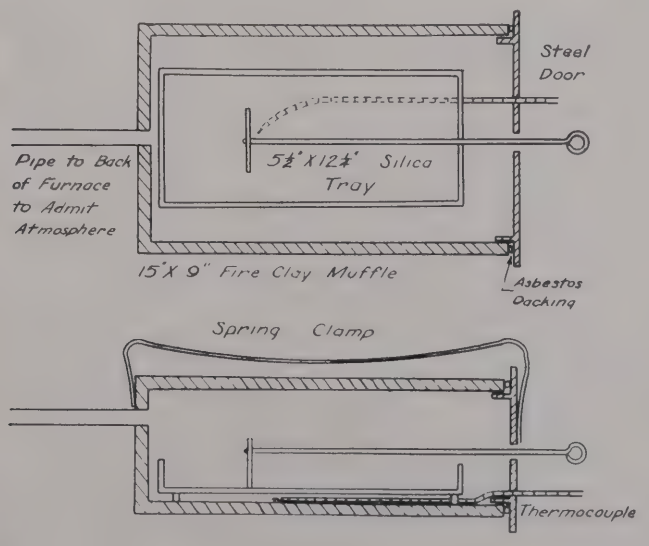

Figure 2

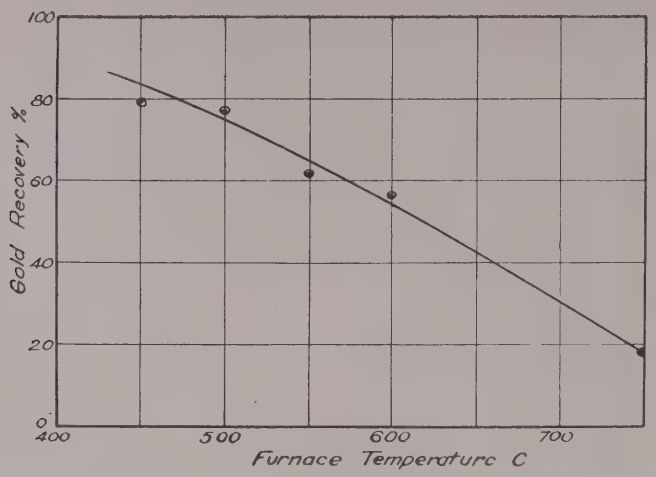

Figure 3.—Effect of furnace temperature on gold recovery from calcines produced by furnace oxidation of distilled concentrate.

Figure 4.—Tailing 0.48 oz. per ton X1,000 Calcine cyanidation residue C-53. Distilled concentrate roasted at 450°C. The pores of much of the iron oxide are extremely small and the walls thin, although partial fusion has occurred to a considerable extent.

stantially constant, while varying the maximum furnace temperature. The results are shown in Table II and comparative gold extractions are depicted graphically in Figure 3.

Analysis of the data in Table II shows that neither the sulphide sulphur nor the sulphate sulphur in the calcine has any appreciable effect on extraction. There is, however, an obvious inverse relation between measured furnace temperatures and gold extractions. It is apparent, also, that low-temperature roasting of the distilled concentrate gives a better extraction of gold (79.8 per cent) than is obtained from the unroasted material (45 per cent), or from the raw ore roasted at a low temperature (67.0 per cent).

Effect of Atmospheric Control on Gold Extraction Effected by Cyanidation

To examine the possibility that a controlled atmosphere is *per se* responsible for improved gold extraction, a series of tests was carried out in which the temperature was kept constant but in which (*a*) a low oxygen-content atmosphere was introduced for different periods of time, and (*b*) a low-oxygen high-sulphur-dioxide atmosphere was introduced for varying periods of time. The results are shown in Table III.

Analysis of the data in this Table shows that neither the sulphide-sulphur nor sulphate-sulphur content of the calcine has any effect on gold extraction if the concentrate has been distilled prior to further treatment. Also, it is quite apparent that time of roast has no appreciable effect and, finally, if the data in this Table be compared with

Table III.—Gold Extractions Effected by Cyanidation Controlled Atmosphere Roasts at 450°C. Furnace Temperature

Test No.	Max. Furnace °C.	Time at Max. Temp. hr.	Furnace Inlet Atmosphere		Calcine		Tailing Au oz./ton	Au Extraction %
			O_2%	SO_2%	S-Sulphide %	S-Sulphate %		
C-50.....	450	2	14	—	17.6	2.7	0.52	77.0
C-52.....	450	4	14	—	4.4	4.0	0.50	78.7
C-49.....	450	7	14	—	3.9	5.0	0.46	82.6
C-63.....	450	4	14	7.4	2.0	4.9	0.54	76.8
C-62.....	450	7	14	7.4	1.1	5.5	0.54	76.0
C-64.....	450	9	14	7.4	0.8	5.2	0.52	78.8

those given for Test *C-53* in Table II, it is equally apparent that extractions are almost identical whether the calcine is roasted at 450°C., with free access of air, or in a low-oxygen atmosphere, or in a high-sulphur dioxide atmosphere. Under the conditions examined in these tests, therefore, the composition of the furnace atmosphere within the limits shown has no direct effect on gold extraction.

Two other factors that have often been considered to have an influence on the extractibility of gold ore are (*a*) the possible 'coating' of gold particles with some compound insoluble in cyanide, and (*b*) the dissemination of gold either in solid solution or occluded by unroasted material or by fused products of roasting. Both factors seem so closely allied to considerations of particle temperature and the effect of fusion during roasting that they have been included in this present investigation.

Effect of Particle Temperature and Fusion on Gold Extraction

From the results given, in which the effects of arsenic content of feed, sulphide and sulphate content of calcine, composition of atmosphere, fine-grinding, and furnace temperatures have been evaluated, there is a strong indication that a high roasting temperature has the most important adverse effect on gold extraction. This suggests that particle fusion with consequent occlusion of fine gold particles may be the controlling factor. This supposition is borne out by the appearance of the calcines (predominantly iron oxide) produced at 450°C., 550°C., and 750°C. furnace temperatures, when these are examined under the microscope (Figures 4, 5, and 6). These are the calcines produced in tests *C-53*, *C-47*, and *C-34*, referred to in Table II, in which the gold extraction was 79.8 per cent, 60.9 per cent, and 16.3 per cent, respectively. The extreme drop in recovery in Test *C-34* is the more striking when considered in relation to the degree of particle fusion evident in the photomicrographs.

As such a state of fusion of iron oxide cannot be reached at 750°C., these photomicrographs provide convincing evidence that either the particle temperature must have

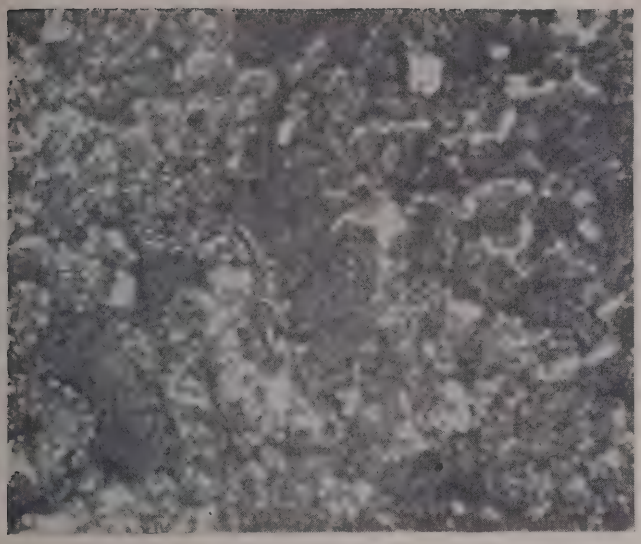

Figure 5.—Tailing 0.84 oz. per ton. X1,000
Calcine cyanidation residue C-47. Distilled concentrate roasted at 550°C. The pore walls are thicker than for C-53 and some grains show almost complete fusion around their boundaries.

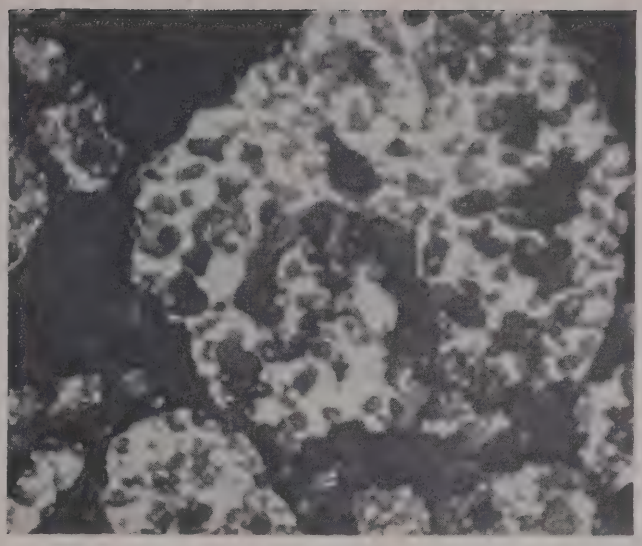

Figure 6.—Tailing 1.58 oz. per ton. X1,000
Calcine residue C-34. Distilled concentrate roasted at 750°C. The pores are much larger than for C-53 or C-47 while the walls are thick and continuous.

been much higher than the measured furnace temperature, or the presence of FeS in contact with FeO must have caused a eutectic lowering of the fusion temperature, as indicated in Figure 12. In order to evaluate the difference between measured furnace temperatures and actual particle temperatures and avoid FeS-FeO eutectic formation, the following technique was employed:

A large sample of distilled concentrate was completely sulphated with sulphuric acid, evaporated, and dried, leaving the iron as ferrous sulphate, with the original particles of gold present. Portions of the ferrous sulphate were thermally decomposed to iron oxide by heating in a tube furnace at 650°, 875°, 1,000°, and 1,100°C. As the decomposition of ferrous sulphate is endothermic, the temperature of the iron oxide particles produced in this manner was at no time higher than the measured temperature of the tube furnace. Photomicrographs of these particles are shown in Figures 7, 8, 9, and 10, and the result of cyaniding the four calcines is given in Table IV.

Comparison of the size and appearance of the particles in Figures 6 (750°C.) and 10 (1,100°C.)

TABLE IV.—GOLD EXTRACTIONS EFFECTED BY CYANIDATION THERMAL DECOMPOSITION OF SULPHATE DISTILLED CONCENTRATE

TEST No.	MAX. TEMP. °C.	TIME AT TEMP. hr.	Fe %	CALCINE S-SULPHATE* %	TAILING Au oz./ton	Au EXTRACTION %	CALCINE COLOUR
C-71....	650	6	42.0	8.0	0.10	96.2	Brick red
C-76....	875	½	52.0	0.9	0.10	95.8	Dark brick
C-102...	1000	½	52.1	0.2	0.48	75.6	Dark red
C-73....	1100	½	52.6	0.2	0.96	50.7	Dense black

*The sulphated material contained no sulphide sulphur.

Figure 7.—Tailing 0.10 oz. per ton. X1,800
Calcine cyanidation residue C-71. Sulphated concentrate decomposed at 650°C. Particles are so fine that they cannot be resolved at 1,800 magnification.

Figure 8.—Tailing 0.10 oz. per ton. X1,800
Calcine cyanidation residue C-76. Sulphate concentrate decomposed at 875°C. Particles are very fine as in C-71.

Figure 9.—Tailing 0.48 oz. per ton. X1,000
Calcine cyanidation residue C-102. Sulphated concentrate decomposed at 1,000°C. Calcine particles have grown due to partial fusion of iron oxide.

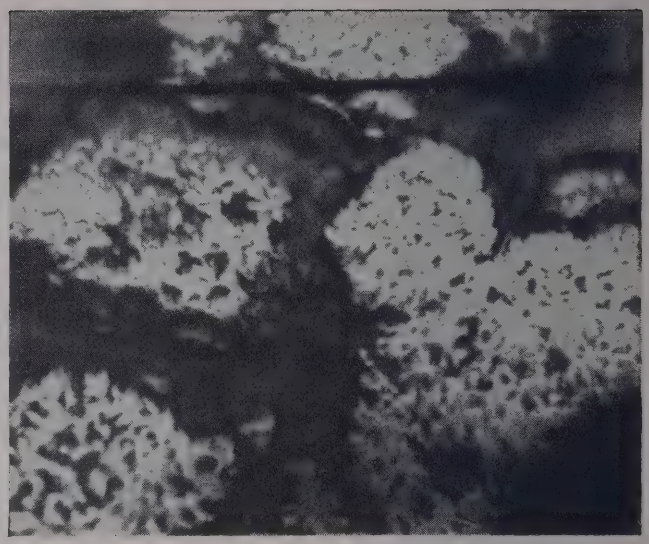

Figure 10.—Tailing 0.96 oz. per ton. X675
Calcine cyanidation residue C-73. Sulphated concentrate decomposed at 1,100°C. Calcine particles have grown and increased in density with increasing temperature.

makes it clear that the individual particles in the sulphide roast of Figure 6 have reached a state of fusion, despite the fact that the measured furnace temperature was only 750°C. Figures 5 (550°C.) and 9 (1,000°C.) show parallel characteristics.

A similar comparison of gold recoveries, as shown in Figure 11, gives added proof that the particles have undergone fusion at a measured furnace temperature which is 400° to 600°C. lower than the temperature required to produce an equivalent state of fusion in the sulphate concentrate. This affords further evidence that conditions of fusion control the extraction of gold.

Two possible explanations present themselves to account for the above facts: (1) the actual particle temperature may be 400° to 600°C. higher than the measured furnace temperature during roasting; or (2) there is reduction in the melting point of the products. The latter statement involves consideration of the FeO-FeS binary phase diagram.

This system is illustrated in Figure 12 and shows a eutectic at 44 per cent FeO with a melting point of 950°C. This is 438°C. below the melting point of FeO (1,388°C.). It is possible that, during roasting, the formation of FeO from FeS may produce a zone of eutectic composition in the particle. Fusion is therefore possible at relatively low temperatures, particularly if compounds of lead, arsenic, or antimony are present. Removal of sulphur from this eutectic during the roast would form a semi-porous structure which would occlude part of the gold.

Examination of Figure 4 (450° C.) shows that, even at this furnace temperature, which is as low as

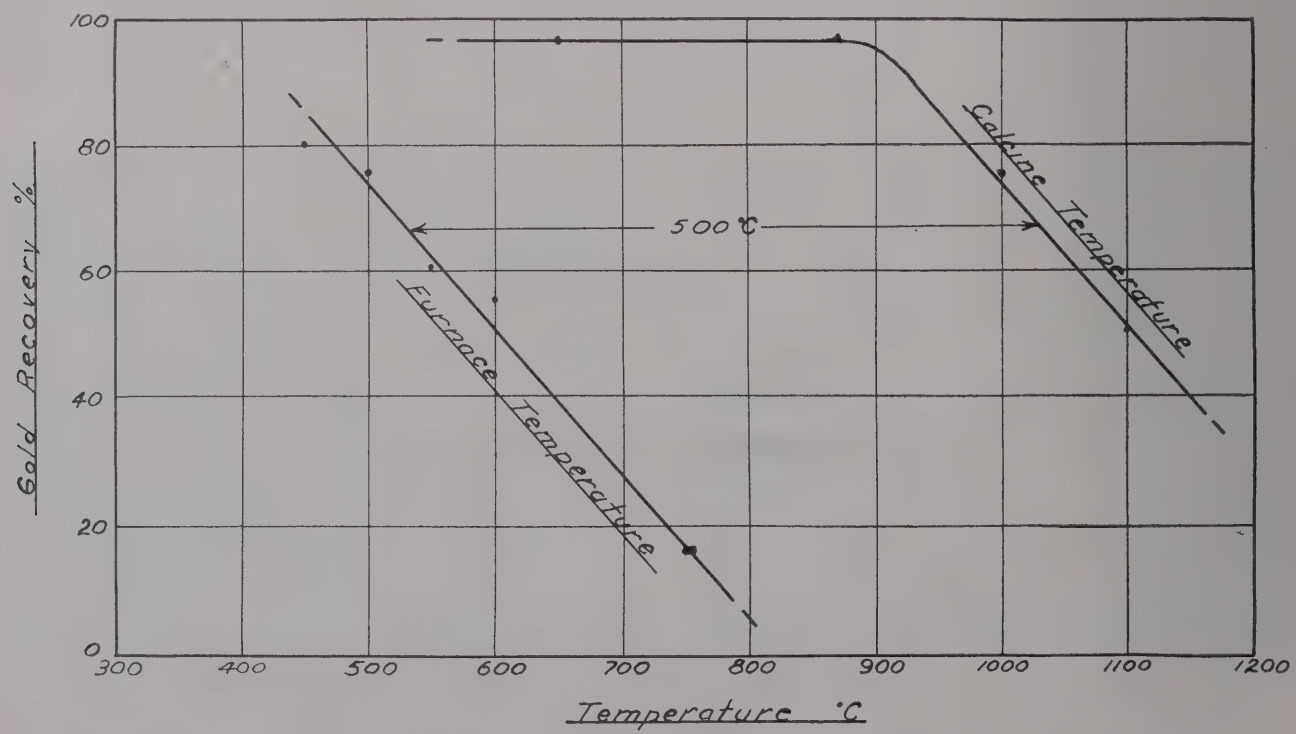

Figure 11.—A comparison of the furnace and particle temperatures during a furnace oxidation of the distilled concentrate.

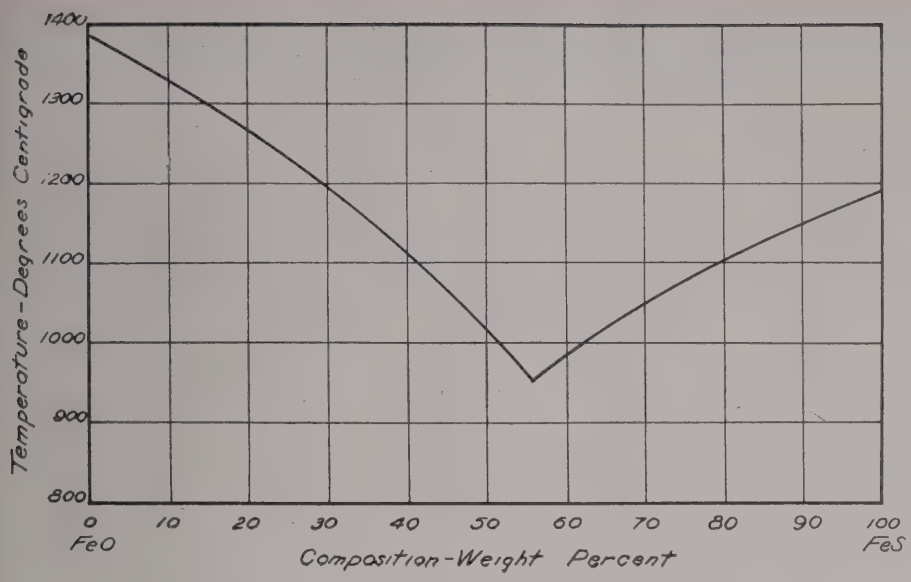

Figure 12

can be successfully operated in practice, there is sufficient particle fusion to render some of the gold insoluble in a cyanide solution, due to occlusion.

This would therefore suggest that a combination of the above two mechanisms, namely, a particle temperature approximately 400° to 500°C. above the furnace temperature and a FeO-FeS eutectic melting at 950°C., accounts for the apparently unavoidable particle fusion and gold occlusion during roasting of these refractory ores.

Development of Procedure for Complete Extraction of Gold by Cyanidation

If the hypothesis be accepted that fusion of particles of calcine with consequent occlusion of gold is responsible for low gold extractions; and were it possible to convert the sulphides to porous oxides under such conditions that the particles could not possibly approach fusion temperatures; then all the gold in the oxidized calcine should be soluble in cyanide.

It was considered that the conditions necessary for such a low temperature oxidation could be established by oxidizing the sulphide in an aqueous medium in which the heat of oxidation is rapidly transferred to the medium, thus preventing the particle temperature from rising above that of the surrounding medium

To accomplish this, the following procedure was developed:

A 200 gm. sample of 'distilled' concentrate was mixed with 500 ml. of 12-normal NaOH solution, and placed in the rocking autoclave shown in Figure 13, which was then heated to 150°C. by an automatically controlled heating element. At this temperature oxygen was introduced and the reaction allowed to proceed for from thirty minutes to one hour, at a pressure of 100 p.s.i.g. The pressure was then released, the autoclave cooled, and its contents discharged to a filter. The solids, principally iron oxide and hydroxide, were washed several times with water, and cyanided in the same way as the roasted calcines. The results of three typical tests are shown in Table V.

Analysis of these results shows that, in the shorter periods of tests *C-106* and *C-107*, the sulphide had not been completely decomposed, but, nevertheless, the gold extractions are higher than any of those obtained by roasting under optimum conditions. Test *C-108* shows that almost all the gold in the product is soluble in cyanide. It is possible, but highly improbable, that the remaining 2 per cent of the gold could be present in solid solution. More likely, the failure to reach 100 per cent is due to the presence of a small amount (0.4 per cent) of unoxidized sulphide, and possibly to

Table V.— Gold Extractions Effected by Cyanidation Oxidation of Distilled Concentrate in Aqueous Medium

Test No.	Autoclave Treatment					'Calcine' S-Sulpdihe %	Au Extraction by Cyanide %
	Temp. °C.	Time hr.	O_2 Press p.s.i.g.	Aqueous Medium			
				Initial NaOH	Final NaOH		
C-106....	150	<1	100	12N	2.5N	7.7	92.4
C-107....	150	<1	100	12N	1.7N	5.5	96.2
C-108....	150	1	100	12N	0.2N	0.4	98.0

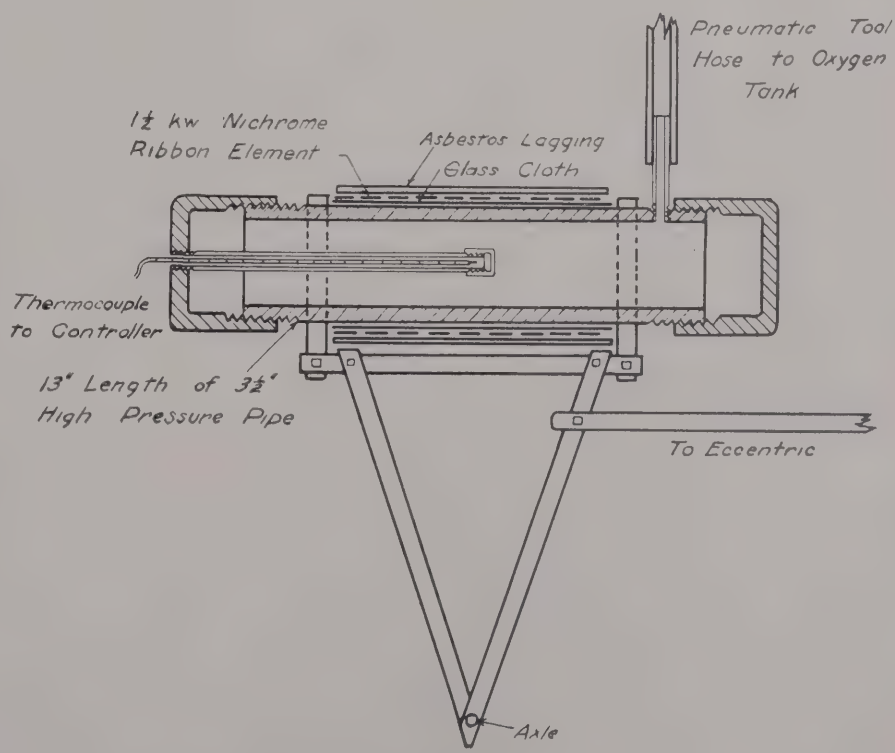

Figure 13

slight adsorption of some of the dissolved gold by the very porous residue.

The results obtained by aqueous oxidation support the hypothesis that the cause of low gold extraction from roasted sulphides is the occlusion of gold particles by fused oxides. The extractions obtained by cyaniding calcines produced at a maximum furnace temperature of 450°C., and the presence of fused particles in these calcines, show that some fusion and gold-occlusion occurs even when a sulphide ore is roasted on a hearth in a controlled gas atmosphere under controlled laboratory conditions. Moreover, as roasting proceeds very slowly when the furnace temperature is below about 450°C., the results of these tests afford an explanation of the difficulty of roasting sulphide ores in a hearth furnace without producing *some* particle fusion. Aqueous oxidation, however, allows no fusion, and thus no occlusion, since the particle temperature cannot exceed the temperature of the medium.

The foregoing results show that the highest gold extraction is obtained by cyanidation if 'refractory' sulphide ores are roasted in such a way that the particle temperature is kept as low as possible. This objective can be approached by one or other of the following procedures:

(1) Keep the measured furnace temperature in a hearth roaster as low as is consistent with the economic rate of roasting.

(2) Hold to a minimum the readily-fusible, slag forming oxides formed during roasting. Lead and antimony would be expected to be the worst offenders in this respect, due to ready formation of silicates and 'glasses' at low temperatures under oxidizing conditions. Arsenic would have some effect, but as As_2O_3 is readily volatile at low temperatures it should not be as serious as lead and antimony.

(3) In the roasting operation, maintain conditions that tend to prevent overheating of individual particles and at the same time keep the formed oxides out of contact with one another. The characteristics of the fluosolid reactor suggest that an operation of this type should provide the best conditions for oxidizing a given ore in a gas atmosphere.

(4) Carry out the oxidation in an aqueous solution with oxygen under pressure.

Conclusions

(1) The refractory gold in cyanide tailings is below the particle size which can be reached by grinding.

(2) The essential agency responsible for low gold extraction from calcined 'refractory' ore is occlusion of gold by fused oxide particles.

(3) The smaller the gold particles in an ore, the more easily they are coated and protected from cyanide by either the minerals of the 'raw' ore or the fused compounds formed during roasting. Thus the more refractory the ore, the more critical the roasting conditions.

(4) The fusion of oxides during the roasting of sulphide concentrates in a hearth furnace is probably the result of: (*a*) the actual particle temperature being from 400° to 600°C. higher than the measured furnace temperature; (*b*) the formation of a eutectic of FeO-FeS having a melting point of 950°C., which is approximately 450°C. lower than the fusion point of FeO.

(5) It does not appear possible to oxidize a sulphide concentrate in a gas atmosphere by conventional methods without causing *some* particle fusion.

(6) The sulphur content of gas-oxidized calcine, whether present as sulphide or sulphate, bears no specific relation to the extractibility of the gold.

(7) Arsenic present in the sulphide concentrate is of secondary importance only: it may decrease extractibility slightly by causing somewhat more extensive particle fusion.

(8) Gold can be almost completely extracted from a 'refractory' arsenical sulphide concentrate by oxidizing the 'distilled' concentrate in an aqueous solution at 150°C. and cyaniding the resulting insoluble oxide residue.

(9) Aqueous oxidation appears to offer a solution to the problem of achieving high recoveries of gold from refractory ores. Although the tests reported were performed on concentrates from which the arsenic had been distilled, there is every reason to believe that aqueous oxidation could effect similar high recoveries from raw concentrates. Such a process would have many advantages, but whether or not it would be economic remains to be determined. Its economic possibilities will be linked with the development of suitable metallurgical equipment and materials.

Acknowledgments

The authors wish to thank the Consolidated Mining and Smelting Company for the ore, and the National Research Council for the financial support of this work.

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Backfilling at Noranda

By F. E. PATTON *

(*Ottawa General Meeting, January*, 1952)

(*Transactions, Volume LV*, 1952, pp. 127-133)

Introduction

AT NORANDA, the strength of the ore and wall-rocks permitted mining in large open stopes. While some of the orebodies were small and could be mined out in one stope, in most cases their size was such that it was essential that ore pillars be left during the first stage of mining. In these cases, backfilling has been necessary for ground control and to facilitate the later recovery of the ore in these pillars.

This paper discusses the development of the backfill and its use in pillar recovery.

Geology

The orebodies at Noranda occur as replacement deposits in a block of rhyolite flows and breccias, bounded on the north by the Horne Creek fault and on the south by the Noranda Andesite fault. Mineralization consists of pyrite, pyrrhotite, magnetite, chalcopyrite, sphalerite, gold, and various tellurides.

The main orebody is a massive sulphide deposit designated the *'H'* orebody. It is elliptical in plan, measuring roughly 400 feet by 800 feet, and is divided into two zones — *Upper H*, extending from surface to the 1,500-foot level, and *Lower H*, from the 1,500-foot level to just below the 3,000-foot level.

Mining Methods

Open stope mining in both *Upper H* and *Lower H* zones has been fully described in previous publications (1). A brief review only of the mining methods will be given here.

A 60-foot vertical transverse pillar runs through *H* orebody from surface to the bottom. The *Upper H* is further divided transversely into 46-foot stopes and 35-foot pillars. In some cases, the pillars have been knocked out previous to filling, with the result that there are some very large open areas.

In the *Lower H*, a 40-foot longitudinal pillar was laid out at right angles to the 60-foot transverse pillar and, parallel to the latter, stopes 60 feet wide with intervening 40-foot pillars were laid out in the four quadrants formed by the two main pillars (Figure 1).

The original method of mining these stopes was by horizontal sub-level benching. This was later changed to long-hole radial drilling, with sectional steel, from sub-level drifts driven along the pillar walls. The final change resulted in the present system of long-hole radial drilling with diamond drills (Figure 2).

The practice is to cut a slot, 10 feet wide, to the full width and height of the stope, at the boundary of ore and wall-rock. Mining then proceeds by blasting into this slot, retreating toward the longitudinal pillar. When mining is finished, and the stope has been drawn empty, bulkheads are placed in the draw-holes above the grizzly level and back-filling is undertaken.

The pattern of vertical pillars and level pillars in *Lower H* left only about 50 per cent of the ore available for open stoping. At present, the open stoping has been largely completed and about 80 per cent of current production comes from the mining of pillars, remnants, and some irregular sections of ore which could not be mined by open stoping.

In the mining of irregular orebodies, a flat-back type of cut-and-fill stoping is employed (Figure 3). Initial development consists of driving chutes and manways to an elevation sixteen feet above the drift rail. The stope floor is cut at this elevation to the stope limits, which are either assay contacts or pillars. After the stope limits have been determined by this first cut, back is taken down over the

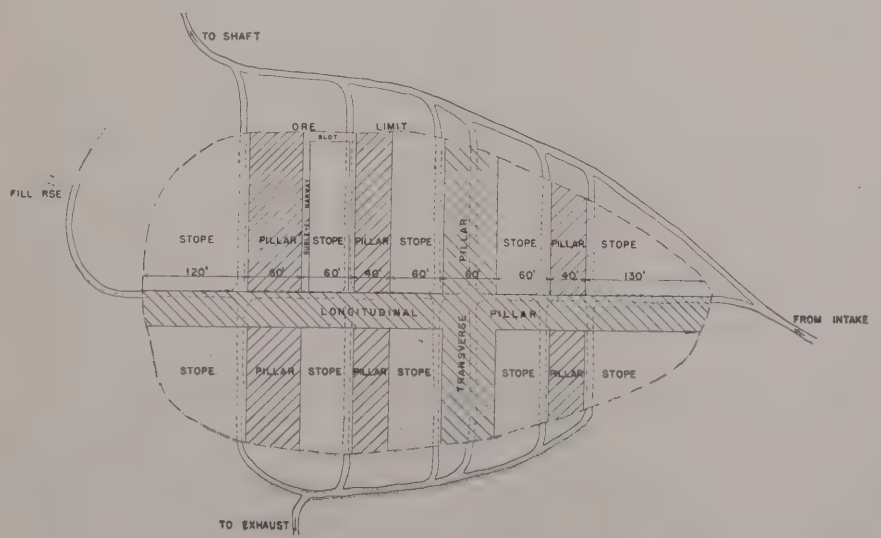


Figure 1.—Stopes and pillars, *Lower H* orebody.

*Chief Mine Engineer, Noranda Mines, Limited.

(1) *Mining at Noranda*, by O. Hall; C.I.M., Bull. March, 1937; Trans., Vol. 40, 1937, pp. 141-164.
Stoping at Noranda, by O. Hall, R. V. Porritt, and A. D. Carmichael; C.I.M., Bull., March, 1939; Trans., Vol. 42, 1939, pp. 133-149.

stoping area so that the resulting undercut has a height of sixteen feet. A square-set mill-hole, with timber-brick lining, is raised over the chute, and cribbed manways over the access raises from the haulage drift, all to a height of eight feet. Eight feet of fill is then placed in the stope and regular mining is commenced.

Sometimes, the width of a stope must be reduced because of ground conditions. This is accomplished by dividing the stope area into two or more panels, one of which will be mined to the final stope height after which the operation will be repeated on the other panels in succession. Before commencing mining of the first panel, backfill is placed tight under the backs of panels left for later mining. An alternative method is to lay out panels radiating from the mill-hole and to mine, one by one, all the panels on each horizon before proceeding with the next lift. In this case, as one panel is finished, a gob fence is placed against the adjacent panel and the completed panel is filled. The gob

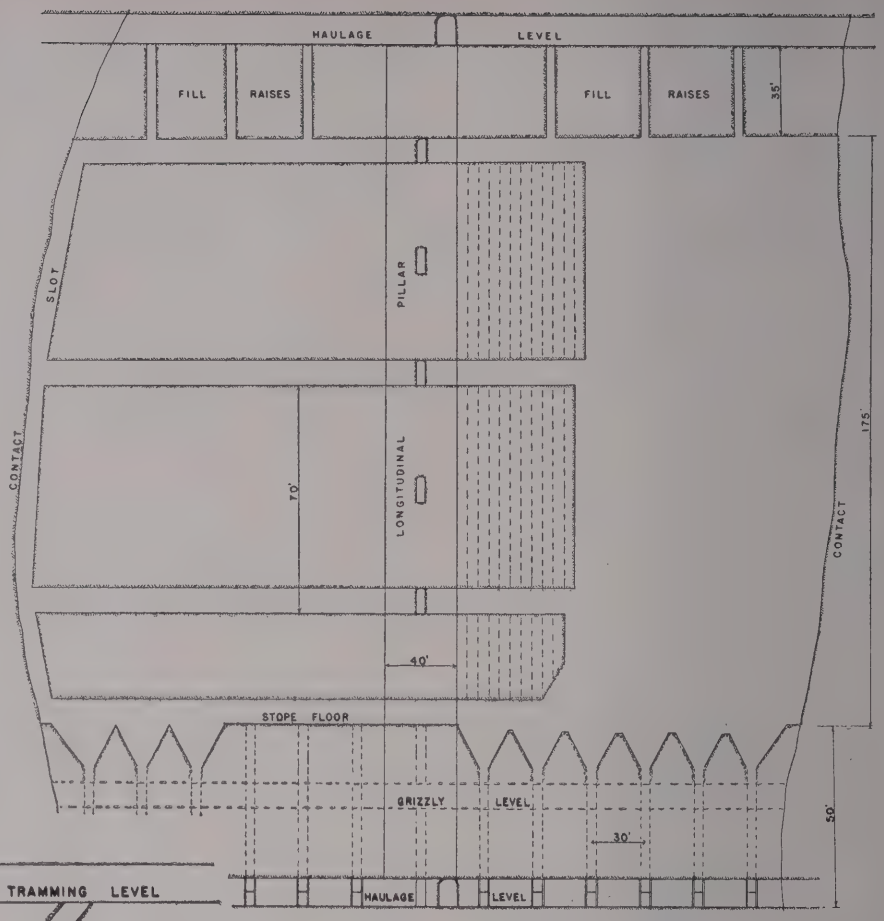
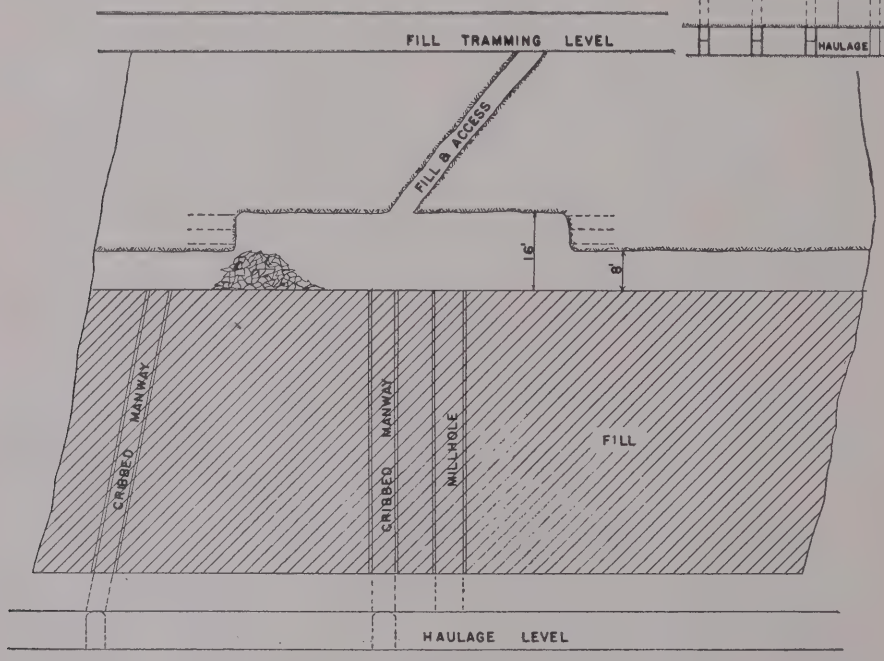


Figure 2.—Longitudinal section, open stopes.



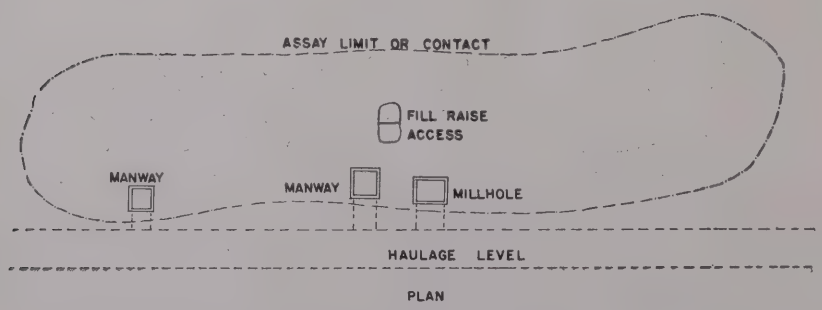


Figure 3.—Flat back stope.

fence is required to retain the fill because the interval between mining adjacent panels is too short for good consolidation of fill.

Rill stoping is used in the majority of pillars in *Lower H* orebody. Rill stopes are laid out longitudinally and mined in vertical slices having a maximum width of 15 feet. (Figure 4). The first slice mined is the one adjacent to the filled open stope. Fill raises are driven in this slice to the haulage level above, and timbered raises and mill-hole are carried, with the stope, adjacent to the next slice. This timber is removed as the next slice is mined.

Broken ore is removed, and fill is placed, by slushing with a 15 h.p. air hoist situated near the mill-hole. A 20° slope has been adopted for the rills as making for maximum efficiency. The only flooring required for slushing is a split lagging floor for a short distance on either side of the millhole.

Backfill Experiments

Under the direction of the late Mr. E. Hibbert, former Manager and Consulting Engineer for Noranda Mines, experiments on backfill were started in 1933.

The overburden on the surrounding countryside being chiefly clay, and sand or gravel not being readily obtainable, reverberatory furnace slag was the obvious material to be used for backfill. In 1933, when experiments on fill started, some 3,000,000 tons of slag were available and there was also the assurance of a continuing supply from the smelter.

Observation of the cementing action caused by the oxidation of fine sulphides in surface dumps, and on floors underground, suggested the possibility of obtaining consolidation in the fill by the introduction of fine sulphides. In considering the type of sulphide to be used it was thought that pyrrhotite would be the best, being more active chemically and carrying less sulphur than pyrite. A supply of pyrrhotite tailing was readily available from the pyrrhotite circuit in the mill.

First experiments were conducted by filling wooden barrels with various proportions of rock, broken slag, granulated slag, and pyrrhotite tailing. Raw steam was introduced into the bottom of the barrels to accelerate oxidation. This experiment was duplicated in the moist air underground, eliminating the use of steam. The results of these tests were used to determine the specifications for the next experiment, in which four wooden bins were used, each holding 100 tons of material.

This, in turn, was followed by a larger experiment in which a small orebody, which had been mined through to surface, was filled with 2 820 tons of material as follows:

40% dump slag, over 1 in. and under 4 in
35% dump slag, under 1 in.
25% pyrrhotite tailing

The tailing used had the following analysis:

Pyrite	6 per cent
Pyrrhotite	56 "
Magnetite	10 "
Insolubles	28 "

The slag was transported to the open stope by truck, each truck being given the correct proportion of tailing before it was dumped. On completion of filling, pipes were driven into the fill so that temperatures could be recorded at different points throughout the mass.

During the first month, the temperature had risen to 190°F. at the top, 400°F. at the centre, and 150°F. at the bottom. These temperatures gradually declined over a six-month period.

The reason for the high proportion of tailing used was to ensure that there would be at least a minimum of pyrrhotite in any segregations of coarse fill. However, due to the high temperature readings, the appearance of sublimated sulphur crystals around cracks in the surface crust, and the occasional odour of SO_2, it was decided to reduce the proportion of tailing in any future experiments and to increase the proportion of fines in the fill to overcome any tendency toward segregation.

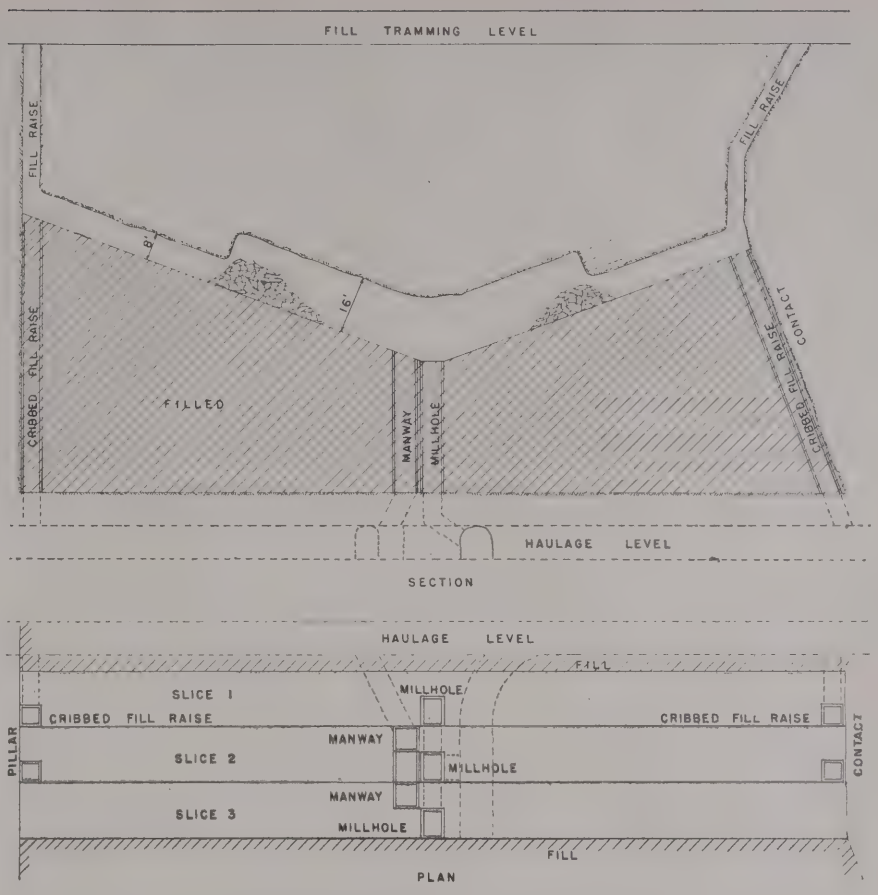


Figure 4.—Rill stope.

The experiments continued and, in the summer of 1935, a small underground stope was filled with 9,250 tons of backfill. This test was conducted in two parts, as follows:

PART *I*

7,500 tons of backfill were poured into the lower portion of the stope. Proportions in this mix were:

50% dump slag, 4 in. maximum
20% granulated slag, ¼ in. maximum
20% granulated slag, crushed to 75% minus 10 mesh
10% pyrrhotite tailing

All material was hauled to a central mixing point and sluiced into the stope with water to prevent segregation of coarse and fine material. The temperature, in the interior of the fill, rose slowly for about five months until it reached 85°F., after which it started to fall. After the fill had set for fifteen months, an inclined timbered raise, 150 feet in length, was driven through the fill by ordinary methods and the fill was found to stand up to expectations. In a few places, near the walls of the stope, the raise went through segregations of coarse fill that was poorly consolidated due to lack of fines.

PART *II*

The upper part of the stope received 1,750 tons of material as follows:

50% granulated slag, ¼ in. maximum
40% granulated slag, crushed to 50% minus 10 mesh
10% pyrrhotite tailing, 200 mesh

The fill was mixed wet in a concrete mixer and poured into the stope to a depth of twenty-five feet. Pipes were driven into the fill and temperatures noted. In six weeks, the temperature in the interior of the fill had risen to 100°F., and in four months it had reached a maximum of 200°F. The outside of the fill remained cool, but it took ten months for the interior to cool. Slight shrinkage of the fill was noted at first, but the original vol-

ume was regained as oxidation proceeded.

After the fill had cooled, a drift was driven through it without the use of any timber (Figure 5). This fill was found to be more compact than that in the lower part of the stope, probably due to the use of finer slag.

Plans for regular backfill operation were based on the conclusions drawn from these experiments, as follows:

(1) Enough fine material should be included to give a high degree of particle contact. However, the fill should be porous enough to allow free drainage of water and some circulation of air.

(2) No more tailing should be added than enough to ensure cementing in any segregation of coarse material.

(3) Future mixes should contain as much fine material as possible but, depending on the availability of slag, some coarse dump slag might have to be used.

(4) In filling large open stopes, raises should be placed at fairly close intervals to minimize the tendency of the fill to build-up in a cone. In such cones, the fine fill is concentrated in the centre and the coarser particles run to the toe.

(5) Tailing should be mixed with the slag by hosing in as a pulp from openings in the side of the stope. Mixing this way would reduce segregation by washing fines down the slope to mix with the segregated coarser material.

(6) Controlled application of water will assist in the oxidation. Rate of oxidation is at a maximum in a warm, moist atmosphere but will be retarded as the fill becomes either dry or excessively wet.

Subsequent fillings were based on these conclusions, although several changes have been made since then. By decreasing the proportion of coarse slag, good fill is obtained with as little as 3 per cent tailing. By sloping open stope fill raises toward the centre of the stope, the coarser pieces of fill, which would tend to segregate, are thrown away from the pillars. No segregation is encountered in cut-and-fill stopes, because the fill is distributed by slushing.

During the first five or six years of backfilling, the operation was carried on at a high rate in order to complete the filling of an accumulation of empty open stopes. During this period, a fairly high proportion of dump slag was necessary to supplement the current production of granulated slag. As this stage of the filling programme neared completion, the reduced rate of filling was such that smaller proportions of dump slag were required. At the same time, an increasing proportion of fill was being used in cut-and-fill stopes, where segregation in the fill was not a problem. Thus, instead of hosing the pyrrhotite tailing into the stope being filled, it became more advantageous to mix the tailing with the stream of slag at control points in the main fill raises.

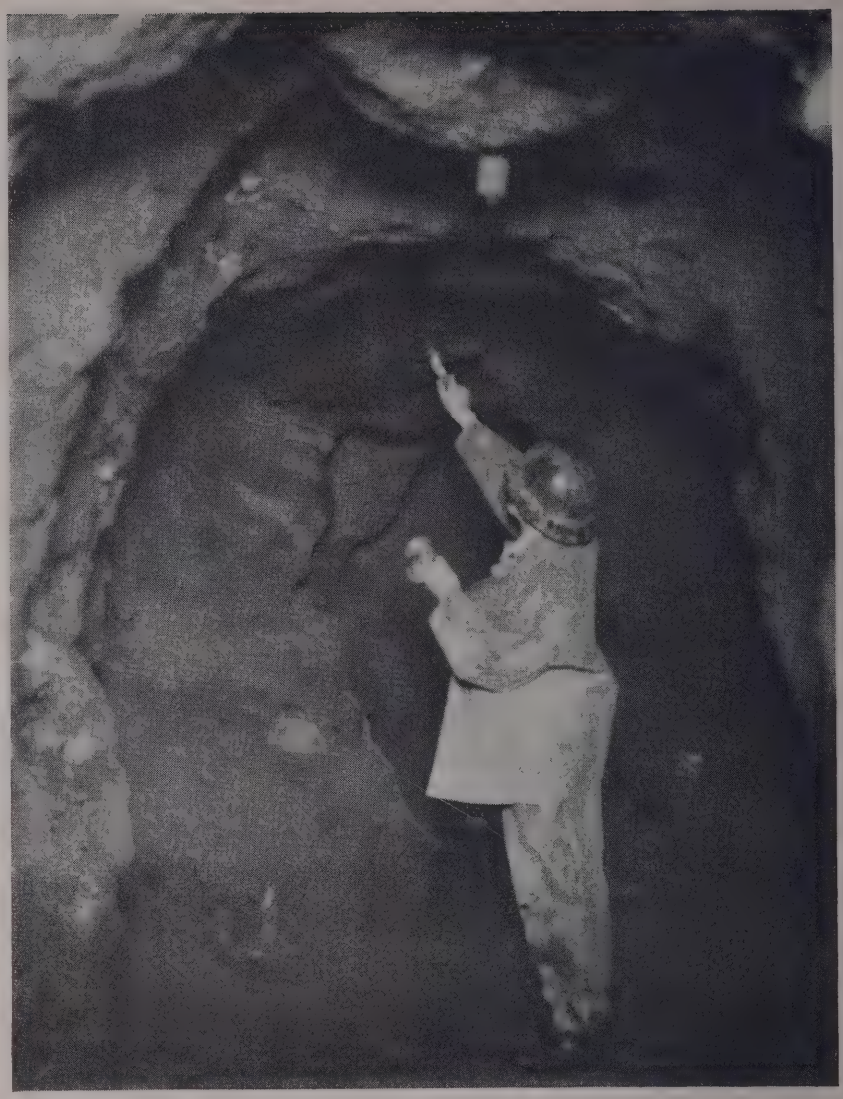


Figure 5.—Showing a drift through the interior of consolidated fill in upper *K* stope.
Hole in background is top of raise coming up from below.

Backfill Operation

The reverberatory furnace slag is granulated by dumping it molten from the slag pots into a tank of water (Figure 6). The tank, or granulator, is of concrete, 20 feet deep, 30 feet wide, and 110 feet long. The mine dump discharge provides an adequate supply of water, the consumption with this method being only a small part of that required for the usual method of granulating with a stream of water.

The granulated slag is removed from the granulator by a clamshell and is stockpiled, and allowed to cool and drain, before being fed underground. Current monthly production of granulated slag is roughly 65,000 tons.

Analysis of the slag is as follows:

Fe	36.5 per cent
SiO_2	38.0 "
Al_2O_3	6.5 "
S	1.5 "
CaO	1.5 "
MgO	1.0 "

The slag is fed underground through a main backfill raise, 7 feet by 7 feet in section, to a control station on the 2nd level and, through a by-pass, to another control station on the 12th level.

Pyrrhotite tailing is de-slimed and thickened to about 60 per cent solids in a small bowl classifier in the concentrator. It is then fed by gravity, at the rate of 20 tons per hour, through a series of connected 1⅞-inch diamond-drill holes to the underground control stations. These holes were drilled from each level. the hole to the level below being drilled in the proper relation to the break-through of the hole from above. At each level both the up and down holes are reamed to 3½-inch diameter to a depth of 4 feet. Extra heavy 3-inch pipe is inserted in the reamed sections of the holes and held in place with rock bolts. The connection is completed by means of heavy-duty rubber pipe of 2-inch inside diameter. At the end of each run of tailing, the hole is flushed with clear water.

At the control points, the slag is fed onto a 42-inch belt conveyor and the tailing into the slag discharging from the belt (Figure 7). The conveyor acts as a measuring feeder, two-foot samples being taken at regular intervals to determine the load. Regular tests are also made to determine rate of flow and density of the tailing. These tests enable the operator to maintain the desired proportions of slag and tailing. The operator has a direct line of communication to the concentrator so that the supply of tailing may be controlled at the source. The percentage of tailing in the mix is adjusted by regulating the flow of slag onto the belt.

Average proportions used at present are:

Granulated slag	72 per cent
Dump slag	25 "
Pyrrhotite tailing	3 "

The mixed fill is passed through a series of raises to the stopes to be filled. It is introduced into the stopes either directly from a branch fill raise, by conveyor belt, or by tramming, the method of introduction depending on the size of the stope and its location in relation to existing fill raises. After the tailing has been added to the fill, the latter must not be left in the raises because of the danger of packing and solidifying, and, consequently, the amount of backfill mixed at one time at the control points is limited to the amount required in a shift.

In cut-and-fill stopes, the fill is distributed by slushing, using *V*-type scrapers (Figure 8) and 15 h.p. air hoists. No slushing of fill is necessary in open stopes until the fill reaches the top of the stope (Figure 9). At this point, the fill is levelled off with slushers to the proper elevation. If tight filling to the back is required it is carried as far as possible by slushing and completed by sluicing with a high-pressure stream of water.

Figure 6.—Molten slag being poured into granulator.

Figure 7.—Slag and tailing mixing point.

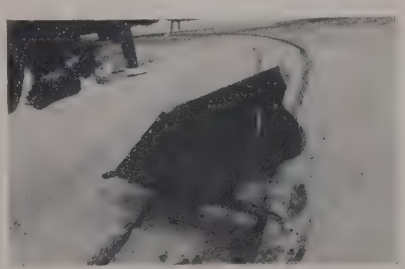


Figure 8.—V-type backfill scraper.

One ton of backfill is required to replace two tons of ore. The present scale of operations requires about 45,000 tons of backfill per month for cut-and-fill stopes. The total demand for fill increases intermittently when open stopes are being filled. A normal open stope requires between 75,000 and 150,000 tons of backfill. Backfill can be supplied to the mine at a maximum rate of 4,000 tons per shift.

Some 13,500,000 tons of backfill have been placed underground since filling commenced in 1938. It may be of interest to note that, of this

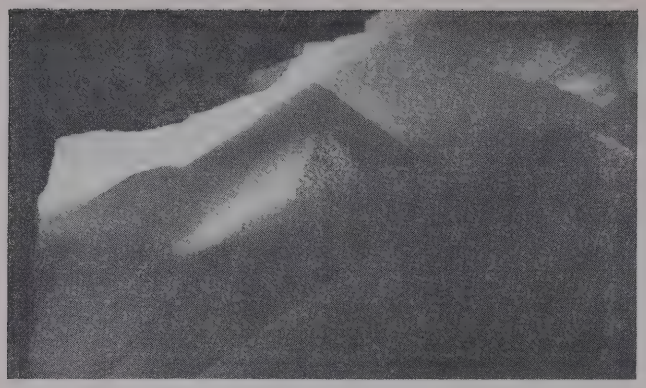


Figure 9.—Filling an open stope.

Figure 10.—Fill wall in stope.

total, 5,000,000 tons was required for two stopes in the *Upper H* orebody which were filled at a cost of approximately $0.07 per ton of backfill.

Oxidation

The heat generated by the oxidation of the pyrrhotite in the fill must be removed by adequate ventilation (1). Difficulty with heat has been encountered only where large bodies of fill have been placed with a high percentage of tailing. Where fill is added intermittently, and in small quantities, as in cut-and-fill stopes, oxidation is completed before the next layer is added, and little difficulty is encountered with high temperatures. It has been roughly estimated that the heat liberated during oxidation of the fill placed in one month is equivalent to that liberated by the combustion of 125 tons of coal.

No definite conclusions have been made as to the cementing action. Polished sections of backfill have shown the presence of an iron oxide on the surface of slag particles. Dilute acid is generated by oxidation and its reaction with the slag may give rise to a cementing product.

Figure 11.—Fill exposed in back of stope.

Oxidation in cut-and-fill stopes is completed in less than three months and in some cases in as little as three weeks. When oxidation is complete, the fill will stand well and no gob fences are necessary to separate ore from fill. Ore can be mined with negligible dilution from adjacent fill. An example of this is the practice, when drifting in ore adjacent to fill, of using a draw cut, drilled to break to the fill. Wall areas as large as 20 feet high and 300 feet long can be opened with little or no caving of the fill (Figure 10). Stopes have been brought up beneath filled areas, leaving fill exposed in the back, without support, over areas 20 by 60 feet (Figure 11). In the final stages of mining one pillar, the ore was completely surrounded by fill and was mined without any support underneath (Figures 12 and 13). The fill sets hard enough that slushing blocks may be anchored to it (Figure 14).

Current costs per ton of backfill are given in the accompanying Table.

(1) *Progress in Underground Air Conditioning at Noranda*, by C. T. Bischoff; C.I.M., Bull., Jan., 1947; Trans., Vol. 50, 1947, pp. 23-35.

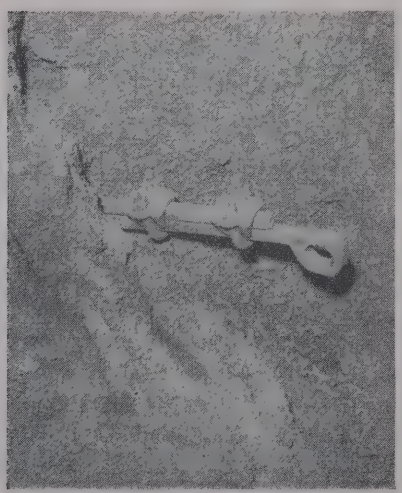


Figure 14.—Slusher eyebolt in fill.

Current Costs Per Ton of Backfill

	Surface Slag	Tailing	Conveying	Placing		Total	
				C&F	Open	C&F	Open
Operation	$0.087	$0.005	$0.080	$0.283	$0.055	$0.455	$0.22
Maintenance	0.076	0.015	0.020			0.111	0.11
Total	0.163	0.020	0.100	0.283	0.055	0.566	0.338

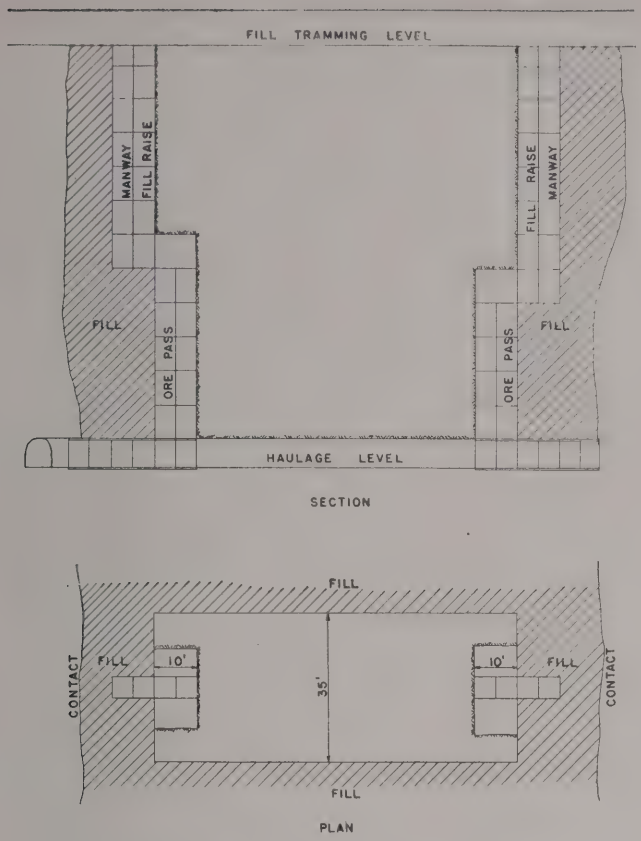


Figure 12.—1153 pillar recovery.

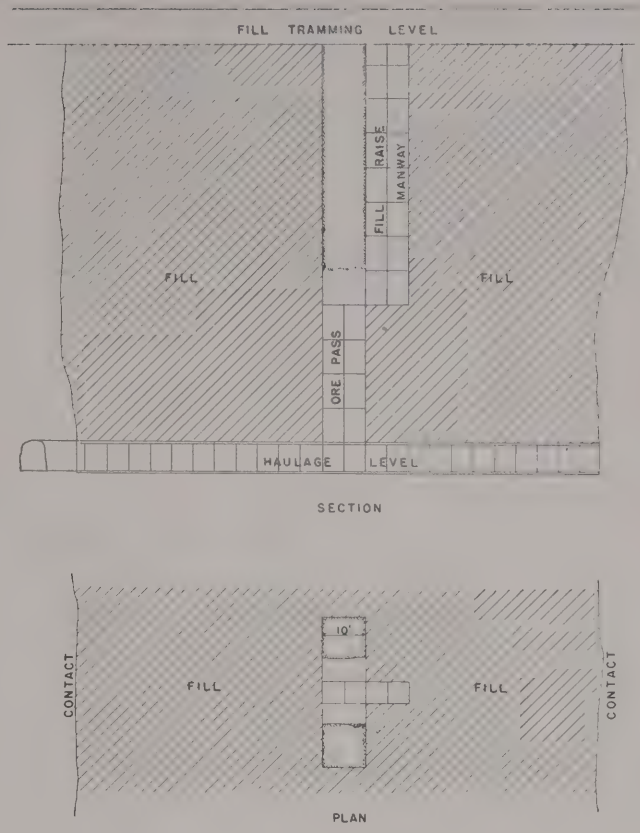


Figure 13.—1153 pillar recovery, last stage.

Summary

Experience with this type of fill may be summarized as follows:

(1) So far as ground support is concerned, the fill effectively replaces the ore and allows maximum pillar recovery under good ground conditions.

(2) Very little timber is required for ground support and practically none for gob fences and mucking floors.

(3) Dilution by fill is negligible.

(4) The cost of this slag fill is very much less than that of any other type of fill available.

(5) The time required for oxidation is short enough that no delay in mining operations is involved.

(6) The extra ventilation required for heat removal is a benefit from other considerations.

(7) The heat and acid water generated by oxidation of the sulphides in the tailing present minor difficulties which, however, are not serious when compared to the advantages gained.

Review of Practices at the Sullivan Mine

Of The Consolidated Mining & Smelting Company of Canada, Limited

By J. R. GIEGERICH *

(*Ottawa General Meeting, January,* 1952)

(*Transactions, Volume LV,* 1952, pp. 134-139)

Introduction

MINING operators in common with other operators are confronted today with two serious problems, shortage of experienced workmen and rising costs. The following review of practices at the Sullivan mine of The Consolidated Mining and Smelting Company of Canada, Limited, outlines certain measures that have been responsible, during recent years, for minimizing the effect of these two problems. The paper is divided into three main sections:

(1) Training programmes and job studies

(2) Improved mechanization and equipment

(3) Large-scale methods of operation

Training Programmes and Job Studies

Job Training

Job training courses were introduced at Cominco immediately after the last war and proved to be a great benefit to the operations. There were three courses given, *Job Instruction Training, Job Safety Training,* and *Job Relations Training.* These were spread over a period of two years and were taken by all operating supervisors and some key operators.

The courses were introduced by a trained instructor from the British Columbia Department of Labour. This man trained twelve of our employees as instructors by giving them an intensive one-week course. Similarity between the three courses was such that the instructors trained in the first one were very readily trained for instruction in the two succeeding courses.

Although the training was mostly limited to supervisors it resulted in:

(1) much better instruction of working personnel,

(2) better appreciation of the causes of accidents and how to remove the hazards, and

(3) greatly improved relations between the personnel concerned. The cost saving of this programme has been intangible. The cost of the programme was low, and indirect benefits have been quite apparent.

Induction School

The induction school for underground employees at the Sullivan mine extends for a period of four days. Since 1946, every employee starting underground has been required to attend this school. Others who are asked to take this course are older employees who have a poor accident record or who do not seem to have appreciated Company policy on routine problems. The number in a class varies from ten to twenty. The induction school is now recognized as making a major contribution toward the improved safety record underground. It has also shown up in better working efficiency.

The course emphasizes primarily the safety rules and practices of the operation. At the start, a historical background of the operation is given, followed by an explanation of general mining terms. From this point, the course is varied by trips to certain points of interest to see in practice what has been discussed in the lectures. Cominco safety rules and B.C. Department of Mines regulations are reviewed and explained. The seriousness of mine fires is emphasized and the procedure to follow in case of such an emergency is outlined. Movies, models, and actual equipment are all used to demonstrate various phases of the operation.

The school is under the direct supervision of the Safety Department, with one member detailed as a permanent organizer and lecturer. He has an assistant who is changed with each class. This assistant is one of the regular operating supervisors, taken off his regular work for a period of a week. The 'supervisor assistant' spends one day with the course instructor before the four-day school starts so that he is well acquainted with the procedure and the part he is expected to handle.

In addition to the two full-time instructors, there is usually an opening talk by the Safety Engineer or a member of the senior staff. During the course, other members of the staff are called on for short talks or merely to drop in as observers. The District Mines Inspector is always called in when a class is conducted during one of his routine visits.

On the last day, the class is shown the Mine Model and a full explanation of it is given by the Model Technician. This is followed by an oral examination on the subjects taken up during the course. Any one not giving a satisfactory performance at this examination is asked to repeat the course at a later date or may be transferred for surface work, or, if results are very poor, may even be discharged. Those passing the test are sent on to the Operating Department for permanent jobs.

Approximately 1,700 employees have passed through this induction school. Fifty different supervisors have been employed as instructors. During the period that classes have been conducted, the accident frequency has been reduced to less than one-third of the previous average. During the same period, direct com-

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pensation costs have been cut in half.

Miners' Training School

Miners' training classes were instituted immediately after the war, when the shortage of miners was acute. The course extends for a period of four weeks which gives the student miner two weeks on each of the two main operating shifts. Four or six men are trained at a time under one supervisor. During the four weeks the men receive training in raising, drifting, and stoping, with the best practice exemplified. Drilling, loading, blasting, scaling, slushing, and scraping are demonstrated and explained. The instructors chosen for the mining school are shift bosses who have shown particular aptitudes during the job instruction courses, and they are assisted in their instruction by the same instructor who is in charge of the induction school.

The personnel for the course are selected from men who have shown promise on general operational work other than mining, such as transportation, piping, and nipping jobs. At the end of the course the trainee returns to his previous occupation and is on call as a spare miner until such time as permanent miner's work is available.

Supervisors' Monthly Meetings

Supervisors' monthly meetings are held for all operating bosses. The meetings cover both accident prevention and operational problems. The first half of the meeting is devoted to 'safety'. The Safety Engineer presides for about one hour and not more than an hour and a quarter. During this meeting, the accident record for the previous month is reviewed. Any supervisor having had an accident involving a member of his crew is called upon to give a full report of measures he has taken to prevent a recurrence. New safety suggestions are reviewed and discussed for approval. A period is also allotted to a safety film or review of mine safety rules or the B.C. Metalliferous Mines Act.

A short recess is called after this first session and the second part of the meeting is devoted to operational problems. The Operating Superintendent presides and he carries on for the same length of time as the first session. The agenda starts off with a review of the previous month's operating performance, illustrating trends, wherever possible, with graphs. Overall performance figures are given for development and production in relation to quotas. Productivity figures are given and compared to target performances set up at the beginning of the year. Cost figures are reviewed and discussed in relation to budget costs.

Schedules of operations for succeeding months are given out at this meeting, with the detailed follow-up left to the foreman. Long-term programmes are outlined so that all supervisors know that advanced planning is being carefully drawn up and so that they realize the importance of their day-to-day work in meeting scheduled deadlines. The meeting usually concludes with a short talk from a member of the staff on some important phase of the operation. Every attempt is made to vary the programmes of these meetings, and at the same time cover the essential routine business.

Trade Apprenticeships

The Cominco Apprentice School constitutes a training programme dating back many years. This incorporates a full tradesman's training in any of the recognized trades which are carried on in Cominco operations. The length of the course is four and a half years plus a six months' improver period. Certificates awarded under the Cominco system are recognized throughout Canada.

Applications for apprenticeships are received from local high school students. The choice of candidates is made by the Personnel Division of the Company in conjunction with the apprentice instructors. Usual requirements are junior matriculation (Grade 12) but, in special cases, lower grades are accepted. Very few tradesmen, excepting for temporary or construction work, have been hired by Cominco during the years that apprentice training has been in effect.

Suggestion Awards

During the war years, various schemes were tried throughout Cominco operations to improve and increase production. One of these schemes was to encourage every workman to suggest ways and means of improving jobs throughout the operation. A Committee, known as the Joint Production Committee, was appointed in each plant to receive and evaluate suggestions received in writing from the employees. The Committee is composed one-half of supervisors and one-half of workmen. The Chairman and Secretary are appointed by Management and meetings are held once a month.

The suggestion is first received by the Secretary, reviewed, and a copy forwarded to the department superintendent. The name of the suggester does not appear during the period of evaluation of the suggestion. The superintendent usually passes the suggestion on to various supervisors concerned with the idea for their comments. It eventually is returned to the superintendent who finalizes these comments and makes a definite recommendation regarding the practicability and economics of the suggestion. He then returns the suggestion to the Secretary of the Joint Committee.

The Committee review these returned copies at the monthly meetings and accept or reject the suggestion. The accepted ones are valued for monetary awards. The approximate scale of the awards is 10 per cent of the annual gross savings. After the suggestion is finalized, the Secretary submits the suggestion and recommendation of the award to Management. On approval, the award is paid and full publicity given to the suggestion through notices placed around the plant.

Awards amounting to approximately $10,000 have been paid by Cominco at the Sullivan mine alone since the inception of the Joint Production Committee in 1941. The largest single award during this period was $500.

Method Studies

Cominco Management first became interested in Method Studies or Work Simplification in 1948. In order to assess the value of introducing Method Studies into our operations, ten staff members were selected from the Trail and Kimberley plants and were given a short intensive appreciation course by engineers from a management consulting firm. The recommendations of this group were favourable and convinced our Management that, in Method Studies, there was a worthwhile tool available for use in elimination of waste energy, time, and material. A second group was then given thorough training in Method and Time Study and a Method Studies Department was formed in 1949 to service the mine and concentrator in Kimberley. Three men were assigned to the work at Kimberley in 1949 but the number is increased as warranted by the demands on the group.

The Method Studies Department is set up as a service department and

work undertaken by the group usually is at the request of the Operating Department. As problems arise, they are turned over to the Method Studies group for examination. The purpose of the study is first explained to the men on the job, by the supervisors and Method Studies technician. It has been found that many of the men on the job have ideas for improving their own working conditions and the efficiency of the job. Where possible and practical, their ideas are implemented without delay, a practice which has a good effect on morale. Those who make worthwhile suggestions are encouraged to submit them for possible J.P.C. awards. During the course of the study, the supervisor directly concerned is regularly consulted and is kept informed of the progress being made. In studying a job, every significant detail possible is observed, timed, and observations recorded.

Upon completion of the study, all data are analyzed and a report giving recommendations is prepared. This report is critized by the junior supervisors and revised where necessary before being submitted 'up to line'. By the time a report reaches the Superintendent, agreement at all levels of supervision has been reached, and usually many of the recommendations have already been carried out.

To date, seventeen problems have been studied by the department. The jobs studied have ranged from one man jobs to those involving a hundred men. Shop layouts, underground transportation, material handling, blast hole drilling and manufacture of items of equipment are examples of studies made by the group.

A study of bit grinding practices was undertaken after the introduction of tungsten-carbide drill bits. Slow processing of the bits appeared to be caused by overcrowded conditions in the shop. After studying the layout with the men and supervisors, a revision of machine location and some minor changes resulted in increased production and a saving of $5,000 annually.

An examination of the underground pipe crew organization resulted in combining certain overlapping jobs and improvement in the flow of information between operators and the pipe crew. Annual savings as a result of this study amounted to $16,000.

Another study was made of the transportation system on one main production level. Through closer scheduling of chute pulling, scheduling of supply handling, and closer planning between the production and engineering departments, a saving of $19,000 annually was achieved.

The foregoing are examples of the work being done by the Method Studies Department. The indicated annual savings that can be made as a result of studies undertaken to date is estimated at $173,000. Improvements can always be made in any job and these improvements can be made at an earlier date by having thorough studies made by trained method study personnel.

Improved Mechanization and Equipment

Transportation and Crushing

Major changes were made in operating practices at Sullivan during 1948. These changes were:

(1) Complete underground crushing

(2) Entirely new transportation system

(3 Introduction of the sink-float process at the concentrator.

(4) The use of float for backfilling in the mine

A detailed paper on these four projects was presented at the 1949 Annual General Meeting of the Institute (1).

With the exception of one underground jaw crusher, crushing operations prior to 1948 were carried out on the surface, with the primary plant at the mine and secondary units at the concentrator. The ore was hauled from the mine to the concentrator by the Canadian Pacific railway. Crushing at the mine plant was done in two stages with jaw and gyratory crushers, the final crushing at the concentrator being done by a Symons and two sets of rolls.

The new crushing plant underground combines these two former surface units and the crushing is done in two stages with a jaw and Symons crushers. A rod-mill replaces the rolls plant. The underground plant is centrally located in relation to ore production and our own transportation hauls the ore from the crushing plant to the concentrator through a new haulage tunnel.

The new haulage system, aside from making large monetary savings, has allowed the convenient return of float to the mine for back-fill. Currently, the weekly production of ore is 50,000 tons for five operating days and from this about 15,000 tons of float. mixed with a small percentage of iron sulphides for cementing purposes, is returned to the mine for backfill. The ore cars are loaded underground from a bin of 15,000-ton capacity. Eight fifteen-ton ore cars can be loaded at one time, while dumping facilities at the mill handle five cars at a time in a rotary dumper. A train of forty cars is hauled by a forty-ton locomotive. Productivity per man shift in the new crushing plant is now double that in the former plants. Transportation costs between the mine and the concentrator show a saving of well over $300,000 per year.

Conveying

The first belt conveyor for ore hoisting was installed at the Sullivan mine in 1944. This unit, consisting of six belts, was 2,600 feet in length and raised the ore 600 feet vertically. The successful operation of the initial installation warranted extension when deeper mining operations were undertaken and an additional 1,800 feet of conveyor was placed in operation in 1951. This latter unit consists of two belts and receives the ore 500 feet vertically below the previous terminus. Experience gained in the first installation was used to make improvements on this second unit. The major change has been in the size of feed to the belt. Finer crushing is now done because of high belt wear caused previously by slabs of quartzite in the jaw crusher discharge. By reducing the size of the crush from six inches to three inches, with a corresponding reduction in slabby rock, it is expected that belt life will be doubled.

Conservative estimates show that conveyor hoisting for the Sullivan has made a saving of 35 per cent on capital costs as compared with shaft hoisting and approximately the same saving in operating costs by taking credit for lateral transportation. Additional units will undoubtedly be added to the present conveyor system and full advantage of new developments in conveyor handling can be incorporated in the construction at that time.

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(1) *Recent Improvements at the Sullivan Operations of The Consolidated Mining and Smelting Company of Canada, Limited,* by E. M. Stiles, E. H. Nagle, and G. J. Knighton; C.I.M. Bull., April, 1950, pp. 212-225; Trans, Vol. 53, 1950, pp. 152-165.

Conveying and transportation are two operations where productivity and costs can be greatly improved by handling a finer product. Delays and abuse to equipment in handling coarse rock have proved to be costly items. Reducing the size of the ore early in the cycle has helped materially to improve costs at the Sullivan.

The crushing unit on the bottom level of the mine, the 2850 level, was placed in operation in 1951 along with the conveyor extension. Previously, crushing had been done at the 3500 level with one jaw crusher. The new two-stage unit comprises a jaw crusher followed by a gyratory. All slabby rock of any size has been eliminated by this jaw-gyratory combination. The ore on the 2850 level is conveyed to the crushing plant above the 3800 level, where it is reduced to 1½ inches in the Symons crushers. Ore from the upper levels of the mine is delivered to this same crusher plant, where it is given its initial jaw crush. It is then joined by the ore from the lower levels and the entire output goes through the Symons crushers.

Slushing

Sluhing ore at the Sullivan is widely practised and is being done primarily because it results in cheaper transportation costs. Early observations of costs indicated that handling ore through short slushers and grizzlies was no more expensive than handling ore through finger raises directly to the loading chutes. The slusher method of handling the ore moreover indicated that a much lower transportation cost could be attained. The result has been that the amount of ore handled through slushers has increased from 25 per cent in 1946 to over 70 per cent in 1951. During this same period, productivity per man shift has increased from 48 to 70 tons per man shift on transportation.

Slushing is never extended over 175 feet except in rare cases where it is cheaper to extend the slusher for small tonnages. Draw-holes normally are spaced at 25-foot centres and staggered on either side of the slusher drift. The standard cross-section of the slusher is seven feet by seven feet, driven five degrees in favour of the load. Slushers are driven as steep as fifteen degrees but they are not considered safe at steeper inclinations. Single hole grizzlies with a 24-inch opening are the usual standard at the delivery point to the raise.

Standard units for scraping are 30, 50, and 60 h.p. double-drum electrical hoists equipped with 42-inch or 60-inch scrapers. Fifteen horsepower hoists and 36-inch scrapers are sometimes used in driving the smaller drifts. Good ventilation is necessary for every slusher operation and the slushers are so laid out that air is delivered at or near the hoist installation and drawn out the far end. Slushers are equipped with spot lights capable of following the course of the scrapers. Atomizers are standard equipment for dust suppression.

Tungsten Carbide Bits

The hard, tourmalinized quartzite, commonly referred to as chert, at the Sullivan has been a major drilling headache ever since it was first encountered. Considerable footwall development is necessary in this formation and, during the war, work in the chert areas was at a standstill because of the shortage of skilled miners and the difficulty of obtaining the proper alloyed grade of drill steel.

The introduction of the tungsten carbide bit has revolutionized drilling in chert. Drill steel that could only penetrate two to three inches per sharpening previously is now replaced with bits that can do two hundred inches of drilling. Advance in standard chert drift headings has improved from half a foot per man shift to over two feet per man shift and contract prices have been reduced by 50 per cent. Earnings are higher and there is less human fatigue. The use of carbide bits in moderately hard ground has been found to give only slightly reduced costs as compared to steel bits, while in the softer ground drilling costs probably are still favourable to the steel bit. Carbide bits have an application for long-hole drilling and will be mentioned under that heading. The standard bit used until recently has been the 1½ inch diameter bit on a 1-inch quarter-octagon shank. One and one-eighth inch diameter powder is used for blasting. Recent test results now favour the 1⅜-inch bit with a ⅞-inch shank, using powder of one inch diameter.

The control of carbide bits and carbide bit steel is under the supervision of a bit technician. His assigned duties include issuing all bits to the miners, who return these same bits whether dull or sharp at the end of the shift. He sorts all returned bits and sends the dull ones to the bit shop for grinding. During the shift he visits those places that are having trouble with drilling and also makes routine visits to all places using the bits. Practically all places using carbide bits are on contract and the contractor is charged $3.00 for any unreturned bits.

Airlegs

The recent introduction of airlegs and light pluggers has improved the performance and practice of driving standard slusher drifts. Their advantage in a standard nine by ten main haulage drift over the jumbo and automatic drifter combination is still questionable.

Standard practice for years has been to drive slusher drifts with stopers, the complete cycle of drilling, blasting, and scraping being done in one shift. The weakness of the stoper method has always been the ragged roof, which is particularly critical in thin-bedded argillite when cutting draw holes. The introduction of the airleg combination has greatly improved this roof condition and is saving considerable expense through the elimination of concrete work. Besides this improvement in roof conditions, the airlegs are giving slightly higher productivity and lower costs and are now considered standard equipment for driving all sub-levels and slusher drifts.

The airleg-light plugger combination has also proven itself superior to the tripod and light drifter for bench stoping operations. This method of stoping, however, accounts for only about 10 per cent of the Sullivan production as 90 per cent of the ore is now drilled with diamond drills.

Large-Scale Methods of Operation

Large Pillar Blasting with Short-Period Delay Caps

The Sullivan orebody above the 3900 main adit level has been mined by a system of open stopes. A very strong hanging-wall allowed stopes to be opened to sizes of 100 feet in width and for two to three times this distance in length. In areas where the vein averaged 150 to 200 feet in thickness, stoping methods were similar to a glory hole bench system with the ore left between draw points being scraped. The original plan laid out for stopes and pillars formed a regular pattern of rectangular blocks with alternate blocks forming stopes and pillars.

Many modifications and changes were necessary, due to ground conditions, as stoping progressed, with the result that a very irregular pattern of pillars was finally left for roof support. The original stoping areas above the 3900 level have now, for the most part, been filled and mining of the pillars has been started.

Individual pillars contain tonnages of ore ranging up to 700,000 tons. Although pillars of this size are usually mined in stages, there are some pillars of approximately 200,000 tons that can be blasted in one stage. The advantages of blasting a pillar in one stage are several. First, the overall dilution is lower; second, it encourages more uniform settling of the hanging-wall; third, it gives a more even grade of ore during extraction; and finally, it makes for a cheaper drilling and blasting operation.

Prior to the introduction of short-period delay blasting caps, shots were held well below 50,000 tons because of the concussion and vibration caused by larger shots when blasted with primacord or the old type delay caps. Today, with the short-period delay caps, the vibration and concussion from a 200,000 ton shot is no more noticeable than that formerly resulting from a 10,000-ton shot. The limiting size of blasts today is not governed by the effects of vibration and concussion but rather by the rate at which the broken ore can be drawn. Fines from the sulphides in the broken ore tend to oxidize in a few months and, unless continual movement can be maintained, heating will occur, causing cementation of the broken ore.

The largest pillar blast to date was fired on April 28th, 1951, when a pillar containing 270,000 tons of ore was blasted. The shot was fired electrically using 80,000 pounds of powder in 3,450 holes. All fifteen delays of the milli-second delay blasting caps were used with the hook-up consisting of 70 series of 50 caps in parallel. Powder factor was 0.295 pounds of powder per ton of ore. Final clean-up has not been made on any of the recent large blasts, so complete comparable productivity figures with previous shots are not available. Preliminary data indicate, however, that much better fragmentation has been obtained with the milli-second delay caps and it has been reflected to date in cheaper slushing costs.

Long-Hole Drilling

In the past ten years, a radical change in stoping methods has taken place at the Sullivan. The change has been from open stope benching and short holes to long-hole drilling from sub-levels. The change was not made with the expectation of lower costs or improved productivity but was necessary because of weaker ground conditions with deeper mining and a change-over to pillar mining on the upper levels.

Results shown by long-hole drilling in sub-levels have proved to be economical, flexible, and safe. The first slots in new stoping blocks are now drilled off with the diamond drill, very few holes being blasted together. Succeeding slices are then blasted by rings drilled six to eight feet apart. Pillar mining in general is done by first extracting the core of the pillar and drawing this ore to allow space for the final blast of the shell. Ore from the final blast is drawn until the dilution from the hanging-wall or stope fill determines the grade cut-off.

The long holes drilled for mining are principally diamond-drill holes with tungsten carbide and jointed steel being used for undercutting stope blocks in mineralized cherty formations. The carbide bit holes have been limited to depths of sixty feet and have been drilled only as plus angle holes with diameters of two inches.

Diamond-drill holes are drilled up to 150-foot depths and vary in diameter from 1 3/16 inches to 6 inches. The small-diameter holes are drilled at any inclination, preferably plus angles, whereas the 6-inch holes have been limited to flat rings. Where applicable, the 6-inch holes have given the cheapest breaking costs. Recent testing data indicate that 1⅝-inch or 2-inch diameter diamond-drill holes will eventually replace the old standard 1 3/16-inch holes.

Diamond drilling is carried out with two men at a machine. A group bonus plan is currently in operation. The drillers are paid a bonus for all footage drilled over a base footage, which is set at the beginning of each month. All drillers and helpers profit equally in this bonus. Average productivity for the past two years has been ninety feet per crew shift for drilling 1 3/16-inch holes.

Open Pit

The Sullivan orebody was discovered in 1892, when four prospectors came upon the outcrop which extended along the surface for no more than 150 feet. The grade of ore at the outcrop was fair as to metal content but extremely complex for smelter treatment. Later exploration of the vein to the north indicated a pinching and lowering of the grade. Exploration down the dip showed a weakening of the mineralization, with only a scattering of the low-grade values disseminated in the argillite.

Early exploration by Cominco after their purchase of the property in 1909 proved up the large and rich orebody farther to the north. It was not until a few years ago that the ore horizon immediately below the original outcrop became of interest. A detailed programme of surface diamond drilling was laid out and, as a result of this, a large low-grade ore zone was proven up. The mineralization was characterized by small disseminated lenses of lead, zinc, and iron sulphides which lacked the usual heavy iron sulphide continuity of the Sullivan vein. High mining and concentration costs were indicated for this ore because of the high percentage of quartzite to be mined.

The area under study showed a tonnage of approximately 2,000,000 tons of low-grade ore. The introduction of sink-float at the concentrator in 1949, along with higher metal prices, made this tonnage of ore very attractive. The mineralized zone was flat dipping, covering an area approximately 600 feet square with a surface cover varying from almost zero to 170 feet in the deepest corner. A detailed engineering study indicated a saving of close to $1,000,000 for an open pit operation as compared to underground extraction.

The open pit project called for the removal of 300,000 cubic yards of gravel and 1,100,000 yards of waste rock. This part of the job was contracted, work being started in October, 1950, and completed in November, 1951.

The gravel overburden was removed by bulldozers in the shallow areas and by shovel and trucks in the deeper areas. The rock drilling has been done with portable wagon drills, with three changes of steel for 30-foot holes using 2- and 2½-inch tungsten carbide bits. Average performance of the drills has been 165 feet per shift and average footage per carbide bit has been approximately 200 feet. Holes at first were sprung but, later, closer spacing was found to be cheaper than springing. Standard spacing of holes has been seven feet. Average powder loading was ⅝ pounds per ton. Split-second delay blasting

caps were used and good fragmentation was obtained in what can be classed as a blocky, hard-breaking ground.

Rock removal was done with two shovels and eight 10-yard Euclid end-dump trucks. One shovel was a five-yard Bucyrus-Erie Electric and the second was a 2½-yard Northwest Diesel. This shovel combination was ideal for the job and allowed for efficient operation of the pit. The average yardage handled by the large electric shovel was 1,400 yards per shift and, for the smaller shovel, 1,200 yards per shift. The waste rock was transferred approximately 1,600 feet on a level haul to a dump area.

The removal of the ore is now being done by Cominco and the rate of production is scheduled for 500,000 tons per year or approximately 2,000 tons per operating day. Drilling, blasting, and ore haulage are very similar to that carried on by the contractors, the one difference being in disposal of the ore. A raise has been driven 700 feet from the main haulage level of the mine to the centre of the pit. The ore is dumped from the trucks into the raise, whence it is delivered to the main transportation level of the mine and joins the underground ore at the main crushing plant.

There is every possibility that, with a successful operation in the present pit area, this type of mining will be extended from the present pit into other shallow areas for final clean-up of low-grade zones.

Summary

Current operating practices at the Sullivan mine which have been instrumental in improving productivity during a period of rapidly increasing costs have been reviewed in this paper. However, there is one fundamental principle which must not be underestimated in any appraisal of mining efficiency and that is the need for good human relations. The cost of most underground mining is still 70 per cent labour and unless everyone on the job appreciates the need for full co-operation in the various changes and innovations made with the object of improved efficiency, results very often may be a great disappointment.

The Occurrence of Uranium in the Lake Athabasca Region*

By S. C. ROBINSON†

(*Annual Western Meeting, Calgary, Alta., October,* 1951)

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Introduction

ALTHOUGH most uranium deposits of the Canadian Shield lie along its western and southern margins, those of the Lake Athabasca region extend easterly along the north shore of the lake and beyond to Charlebois lake. The western end of this belt of deposits, in the vicinity of Goldfields, is 80 miles from the Palaeozoic contact, and its eastern end, at Charlebois lake, is 240 miles from the nearest Palaeozoic rocks.

Deposits in this belt may be grouped in four districts, which are, from west to east:

(1) The Goldfields district, an area approximately 20 miles square, centred on Beaverlodge lake and extending eastward from St. Joseph's point to Moose channel.

(2) A small area in the vicinity of Sucker bay, which is about 10 miles east of Fond du Lac.

(3) An area including Middle lake to the northwest shore of Black lake at the Nisto mine.

(4) A fairly large area extending northward from the eastern end of Black lake, through Charlebois lake to Spreckley lake.

History

The region has stimulated interest in mining circles since the reconnaissance survey of J. B. Tyrrell of the Geological Survey of Canada in 1893, who reported concentrations of iron along the north shore of lake Athabasca. The deposits were staked and re-staked up to the early 'twenties' but are too lean to form ore. Interest was next awakened by the fabulous enterprise of H. Dardier who, backed by British interests, brought out a Calyx shot drill in attempts to prove up nickel deposits in the vicinity of Sucker Bay between 1914 and 1919. The drill, boilers, shot, and over 400 feet of 5½-inch core remain as a memento to his enterprise. Local Indians, incidentally, have discovered that manganese-steel shot is harder on their guns than it is on their quarry. Discovery of copper and gold in the Goldfields and Sucker Bay districts in 1934 inaugurated the next boom in the region, during which the Box and Athona gold mines were developed and hydro-electric power was brought in from Wellington lake. The prospecting boom ended with discovery of gold at Yellowknife, N.W.T., and mining came to an end in 1942 as a result of the low grade of the ore and the changed economy of the war years.

However, during development of the Box and Nicholson properties, uranium minerals had been recognized, and the wartime demand for this metal led, in 1944, to a joint examination of the Nicholson property by Crown-owned Eldorado Mining & Refining (1944), Limited, and the Geological Survey of Canada. As a result of this examination, the Company launched an intensive prospecting programme in 1945 which has led to the development of the Martin Lake, Eagle, and Ace mines. With the announcement this year (1951) that the Ace mine will be brought into production, the Goldfields district is being established again as a mining area.

In 1948, the Federal Government repealed a ban on public prospecting for uranium, and shortly thereafter the Government of Saskatchewan established the concession system of leasing prospecting rights in parts of the Goldfields, Sucker Bay, and Black Lake districts. Active and well-directed prospecting both of these concessions and of other ground has uncovered several promising deposits of uranium. Last summer, upwards of fifteen diamond drills were in operation in the Goldfields district, and four at Charlebois Lake. At Goldfields, five shafts have been sunk on uranium properties and two more are being collared. In addition, four adits have been driven in the Goldfields district and two at the property of Nisto Mines, in the Black Lake district.

Geology

Uranium deposits of the belt extend well into the central part of the Canadian Shield, but are parallel to the northern boundary of a large area of Proterozoic rocks of the Athabasca series. In the Goldfields district, outliers of these Athabasca rocks rest unconformably on the Tazin group, probably Archaean in age. The uranium deposits of the region occur principally in the older Tazin rocks, but in the Goldfields district they also extend upward into the Athabasca series. The belt as a whole, therefore, lies at a horizon near the Proterozoic-Archaean unconformity.

There is no evidence of major structural displacement parallel to the main direction of the belt. Evidence available suggests merely that the Tazin rocks to the north dip regionally beneath Athabasca rocks to the south.

Three major faults that can each be traced for some miles are well established; these are the Black Bay fault, the St. Louis fault, and the Black Lake fault. The entire region, with the exception of the Charlebois Lake area, is traversed by very numerous minor faults and tension fractures and by some small shear zones. Moreover, there is good evidence that movement along the St. Louis fault has taken place at more than one period. In general, efforts to establish a definite fault

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pattern have not yet proven successful, except in very restricted areas. Throughout the region, structural control (mainly by faults, shear, and breccia zones) is the major factor in determining the location and extent of deposits of uranium; but there does not appear to be any consistent association of these deposits with any one specific direction of structure. At Charlebois lake however, the deposits occur along the contacts between granite and metamorphic rocks and lack evidence of any other structural control.

Types of Deposits

Uranium deposits of the region may be divided into two groups: the epigenetic or vein deposits, and the syngenetic or pegmatite and pegmatitic granite deposits. The principal epigenetic deposits are grouped at the centre of the Goldfield's district and at Black lake. The important syngenetic deposits are at Charlebois Lake, with less important ones at Sucker bay and others forming a perimeter around the epigenetic deposits of the Goldfields district.

At Goldfields, deposits of both types are spatially intermingled, and in a few instances there is some evidence of transition, particularly in the area of the Black Bay fault and farther to the west and north. At Sucker bay, both epigenetic and syngenetic deposits occur, but with one possible exception the deposits of the Charlebois Lake district are apparently syngenetic. Throughout the region as a whole, however, it is clear that a close spatial association exists in the distribution of both types of uranuim deposits.

Structurally, three types of epigenetic deposits are recognized. The first, and by far the most common, comprises small, sharply defined, tensional fractures in which deposition is clearly due to vein-filling. Very rarely are these fractures of sufficient size to constitute an orebody in themselves. Where, however, several of them occur closely spaced *en échelon* (commonly related to a major fault), or otherwise in a zone, it may be possible to mine them collectively or as individual stopes from a series of cross-cuts.

Epigenetic deposits are also found in a few shear zones, some of which attain lengths of some hundreds of feet. Such shear zones rarely attain widths in excess of five feet, and in them the uranium content varies greatly over short distances.

In a few instances, uranium minerals have been deposited in fracture or breccia zones along the footwalls of major faults. In these, considerable lengths and thicknesses of uranium ore may exist, as at the Ace mine.

Syngenetic deposits of uranium occur in pegmatitic or granitic zones whose length usually conforms to the direction of foliation or bedding of the host rocks. As Mawdsley (1951) has demonstrated, the pegmatitic zones of Charlebois lake almost everywhere follow the contact of granite bodies with metamorphosed sediments. These syngenetic deposits very rarely show sharply defined contacts with the host rocks. Usually they are banded and transitional to the country rock. Distinction between those bodies that are designated as pegmatites and others designated as granites is largely arbitrary, and is based on such factors as grain size, shape, and extent of the zone. The pegmatites are usually banded, the bands being composed of leuco-pegmatite, biotite concentrations, feldspar-lean and quartz-rich rock, and bands of altered rock. The granitic bodies are made up of coarsely crystalline quartz and feldspar, often with chlorite as the dominant mafic mineral.

Within the pegmatites of the Charlebois Lake district, uranium values are distributed in lens-like shoots within the pegmatite zone. Values usually are restricted to certain bands in the pegmatite and may be continuous along the strike over distances up to 300 feet. In other districts, distribution of uranium in pegmatite is erratic and patchy. The uranium content of the granites is usually lower than in the pegmatites but is more evenly distributed. Most granites throughout the region show detectable radioactivity, but those bodies in which it is unusually high are rarely larger than 500 feet in greatest horizontal dimensions. It is improbable that the uranium content of even the most radioactive of these small granite bodies will attain ore grade under present conditions.

In a few instances, minerals normally epigenetic in origin are found in the pegmatites and granites. Their deposition does not appear to be clearly attributable to any structural feature, but they occur in what were possibly miarolitic cavities, or as pods that appear to be almost transitional to the rock in which they are found.

Influence of Host Rocks on Mineral Deposition

In many instances, some of them very striking, where a fracture system extends through rocks of markedly differing composition, it has been found that uranium and allied epigenetic minerals occur only in one rock type and not in the other. In such cases, deposition has taken place in the mafic rocks — amphibolites and other iron- and magnesium-rich rocks — andesites, and carbonate rocks, and is virtually lacking in the interbedded quartzite, arkose, or granitic gneiss. In general, rocks least resembling granite in composition are favourable hosts to uranium deposition. There are, however, numerous promising deposits in both quartzite and granitic gneiss.

The Association of Uranium with Red Alteration of Wall-Rocks

Epigenetic deposits of uranium are usually marked by red alteration of the wall-rocks; such alteration is lacking or insignificant in amount in syngenetic deposits. It is a fact, however, that red granites are more radioactive than are the grey granites.

Red alteration of wall and host rocks may extend over tens of feet or may be restricted to a few inches in which the uranium occurs. Moreover, red alteration may flank fractures containing no uranium minerals and, conversely, in a few occurrences of high-grade pitchblende in tight fractures, red alteration is lacking. However, red alteration of the host rock is certainly the best visible guide to the presence of pitchblende. A bright, brick-red colour may be broadly indicative of higher than average radioactivity, but concentrations of uranium are marked by a dark red colour verging on dark reddish-brown. The colour is due to disseminated dusty hematite, which becomes darker where dissemination is more intense. However, in the darker areas there is usually an increase of both carbonate and chlorite in the rock.

Although the red colour is due to hematite, in many cases it is accompanied by feldspathization and/or silicification of the rock. Where this type of alteration is intense, the foliation and texture of the original rock are obliterated and the resultant rock is red, with a cherty to vitreous texture. In some instances, notably at the Ace mine, K. R. Dawson (1951) has

found that the red altered rock is composed almost solely of plagioclase feldspar impregnated by hematite.

Mineralogy

Minerals found in the two types of deposit are markedly different and will be discussed separately.

In the epigenetic deposits, pitchblende, the cryptocrystalline variety of uraninite, is the only important uranium-bearing mineral. Thucholite, a hydrocarbon containing some uranium, is the only other hypogene mineral recognized and occurs sparingly in a few deposits. A substance originally described as uraniferous opal is now known to be hisingerite containing minute particles of pitchblende.

Basically, the mineralogy of the epigenetic deposits is simple and consists of pitchblende with hematite, calcite, and quartz. In this oxide association, sulphides, usually in minor amount, are yet remarkably persistent in occurrence. Of these, pyrite is most common, but galena and chalcopyrite, usually very finely disseminated, are more closely associated with the pitchblende. Locally, at Nicholson and Fish Hook bay, cobalt, nickel, and iron arsenides occur with pitchblende. Pitchblende deposits lying in a belt extending northward through the western shore of Beaverlodge lake contain a group of copper and lead selenides associated with native copper. Other minerals found locally in the pitchblende deposits are: native gold, in several deposits and in one of them associated with platinoid-group elements; an iron vanadate, as yet not named and known only from two deposits in the Goldfields district; tiemannite, a selenide of mercury, from one deposit; and octahedrite, from two or three deposits. Manganese and tungsten have been detected in ore analyses, but the minerals containing them have not yet been identified. In addition, there are quite a large number of supergene minerals containing uranium, vanadium, copper, cobalt, nickel, and selenium that are principally of academic interest. They are locally useful to the prospector due to their bright yellow, orange, blue, and green colours.

In many of the vein deposits, textures indicate that deposition of the metallic and gangue minerals took place in open fractures. Comb quartz is common and even calcite shows some tendency toward comb texture. Vugs, too, are occasionally found, and many of the minerals lining these are in well terminated crystals that have been coated over by late calcite. The pitchblende commonly has a colloform habit, and often coats fragments of altered wall-rock. However, a later generation of pitchblende is massive and even euhedral in a few deposits. In several occurrences, this massive pitchblende, examined megascopically, has a distinct bronze to brassy lustre and, under the microscope, it is seen to consist of an extremely intimate intergrowth of pitchblende and calcite in which sulphide grains, usually of the order of a few microns in diameter, are disseminated. These sulphides may be pyrite, chalcopyrite, or galena, or a mixture of any two or all of them. The association is so intimate as to suggest contemporary deposition, in marked contrast to the colloform pitchblende, which contains sulphides only in syneresis cracks.

Paragenesis of these deposits is difficult to assess with confidence. As usual, the more work that is done, the less certain one becomes of the order of deposition. However, a general sequence of events appears to be as follows:

(1) Introduction of hematite, feldspar, and silica into the wall-rock, followed by brecciation or jointing.

(2) Infusion of this brecciated rock by calcite, with deposition of colloform pitchblende and cementation by feldspar and chlorite.

(3) Further brecciation both of the wall-rock and of colloform pitchblende, followed by deposition of calcite, quartz, massive pitchblende, and sulphides. The arsenides were probably deposited early in this phase.

(4) Cementation of open fractures by calcite, associated locally with selenides and native copper.

(5) Supergene alteration of the deposits, which locally penetrates to a depth of 300 feet.

In the syngenetic deposits, the principal uranium mineral is uraninite. Locally monazite, zircon, titanite, and apatite, all slightly radioactive, are also present. Of the complex uraniferous multiple oxides containing titanium, tantalum, and columbium, which are common in pegmatites of southern Ontario and Quebec, only fergusonite has been recognized in the Athabasca region, and that from only one deposit. A point of interest is that thorium occurs in relatively small amount as compared with uranium.

In the Goldfields district, the redioactive pegmatites and granites are composed principally of potassic and plagioclase feldspar with quartz. Accessory biotite and muscovite are in small amount and commonly in local segregations. Chlorite is the principal mafic mineral of the radioactive granites. Minor hematite, and some magnetite, are commonly present. The occurrence of pods of pitchblende, sulphides, and quartz in these syngenetic deposits has already been mentioned.

The pegmatite deposits of Charlebois Lake district are more complex. They have no sharp boundaries and are variably banded, so that it is often difficult to decide within a matter of some feet where the contact between the metamorphosed sediments and the pegmatitic zone should be drawn. In general, however, the uranium favours the distinctly pegmatitic facies rather than the transitional bands.

The pegmatite is made up of quartz with both potassic and plagioclase feldspar; in some bands quartz may comprise 80 per cent of the rock, in others as little as 20 per cent. Biotite is the principal mafic mineral and varies considerably in its distribution. It is found disseminated fairly uniformly in parts of the pegmatite as individual books up to an inch in diameter. Elsewhere it occurs in solid bands of coarse-grained biotite and, again, making up finer grained quartz-biotite bands. In some zones, calcic pyroxene and/or amphibole take the place of the biotite.

Of the accessory minerals, molybdenite is the one most commonly associated with uraninite. Apatite is commonly present and locally forms a large percentage of the rock, not in coarse crystals but in masses of fine needles, usually with biotite and molybdenite. Other accessory minerals are zircon, titanite, magnetite, beryl, pyrrhotite, and chalcpoyrite.

Uranium distribution, notoriously erratic in most pegmatites, does show some consistency at Charlebois lake, over lengths of up to 300 feet in some zones; but in other occurrences, although there is no lack of uranium, it is restricted to small patches. Generally, there is some association between uranium and zones in which book-biotite occurs, but the solid bands of biotite and the quartz-biotite bands are not commonly radioactive. In zones

where a blue-grey quartz predominates, uranium concentration is often relatively high, as also in zones where apatite is concentrated. One successful prospector in the district believes that uranium deposition is greatest where the pegmatites have invaded highly calcic sediments.

Age of the Deposits

Age determinations on ores of the Goldfields district and based on the Pb/U ratio are listed by Ellsworth (1950) and show a range from 5.6 to 14.1 $\times$ 10^8 years. More recently work by Collins, Freeman & Wilson (1951) based on the Pb^{206}/Pb^{207} ratio gives a range in age, as listed by Lang (1951), from 14 to 16.8 $\times$ 10^8 years. The age determinations for the uranium in the Goldfields district thus allow a very wide range, from very early Precambrian (the maximum for which, anywhere in the world, is 22 $\times$ 10^8 years) to the base of the Cambrian (5.5 $\times$ 10^8 years). A number of the pitchblende deposits occur in fractures in what appear to be Proterozoic rocks. For the present, interpretation of the determined ages must await further mineralogical and geological data.

Origin of the Deposits

The association of the epigenetic and syngenetic deposits in spatial distribution has been mentioned previously, as have also the few instances in which there is some evidence of transition between the two types. Further, the appearance of both the pegmatites and granites suggests that they are products of alteration of the host rocks, almost certainly by addition of material from outside. Some parallel in their formation may be drawn with the intense feldspathization of the wall-rocks of some of the epigenetic deposits. It is suggested, therefore, that the two types may be closely related genetically. The great spread in ages, as determined to date, makes any hope of correlation by age determination remote. As Christie (1949) has pointed out, there are no intrusive rocks in the Goldfields district younger than the Athabasca sedimentary and volcanic rocks, in fractures in which some of the pitchblende deposits occur.

It is believed that more than one period of mineralization is represented in many of the uranium deposits of the Goldfields district. Colloform pitchblende is among the earliest of the minerals to have been deposited. It can be fairly decisively proven that the massive type of pitchblende, with associated sulphides, is younger than the colloform type. Moreover, there are a number of sulphide and gold deposits in which pitchblende does not occur. This is true also of some selenide deposits. As a possible working hypothesis of origin of the epigenetic deposits, it is suggested that, originally, uranium was deposited in an oxide environment with hematite and calcite, probably at a somewhat later stage than the formation of the syngenetic deposits. It can be shown decisively that the colloform pitchblende was fractured and cemented in a matrix containing calcite, quartz, sulphides, and massive pitchblende. This may well represent a separate and much later phase of mineralization. Possibly the massive pitchblende is due to solution of some colloform pitchblende and re-deposition by solutions carrying the sulphides. Selenide mineralization is probably a late phase of the sulphide deposition, as there is no evidence of a break in deposition except the lack of seleno-sulphides. For this hypothesis to stand, however, an earlier epoch of mineralization, during which gold and sulphides, dominantly those of iron and zinc, were deposited at the Box and Athona mines, must be recognized, as Christie (1949) has reported evidence that pitchblende at the Box mine is restricted to a set of fractures that cut those containing gold and sulphides.

Several geologists have formed the opinion that many epigenetic pitchblende deposits are of epithermal type, formed at shallow depth and low temperature. Such deposits usually do not extend to depths much in excess of 1,000 feet. Diamond drilling in the Goldfields district has encountered pitchblende intersections with the same relative frequency at depths close to 1,000 feet as it has at depths of 100 feet. In the lack of information at depths in excess of 1,000 feet it may be assumed that pitchblende will continue at depth. This is borne out by the vertical and horizontal extent over which pitchblende has been discovered in the Goldfields district. At Port Radium, pitchblende is known to occur over a vertical range of at least 1,300 feet (Buffam and Gillanders, 1951). Moreover, it is of particular interest to note that Haycock and Graham (personal communication) have discovered evidence at Port Radium indicating that concentration of uranium is associated with quartz, whose indicated temperature of deposition is usually in excess of 400°C.

In the Goldfields district, it would appear that the original pitchblende was deposited at relatively high temperatures and that, therefore, there is no reason to believe that it is restricted to a relatively shallow vertical range.

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Well Logging and Testing Techniques Applied to Reservoir Analysis

By R. POT *

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(*Transactions, Volume LV,* 1952, pp. 144-147)

Introduction

THE PURPOSE of this paper is to discuss briefly the main logging and testing devices presently at our disposal, to point out some of their limitations, and to suggest possible improvements of the techniques.

Some of the more important devices are those that are used in:

(1) Coring
(2) Formation testing
(3) Electric well logging
(4) Radioactive well logging
(5) Measuring of bottom hole pressures
(6) Sampling of reservoir fluids

The first four of these procedures are concerned with the well while it is in the process of drilling. The last two are carried out as soon as possible after a well has been brought into production and at intervals during its life. My remarks will be concerned mostly with certain aspects of formation testing.

The reservoir engineer is dependent on the information derived from these operations in order to obtain a clear picture of the conditions in a reservoir and to enable him to assess and forecast its future behaviour.

Since any calculation is no better than the data on which it is based, not enough emphasis can be placed on the necessity for the testing and logging equipment to be of the highest quality and for the analysis of the results obtained to be thorough.

With regard to quality of equipment, it may be said that practically all service companies, through their research departments, are continually effecting improvements. The Canadian oil industry receives the full benefit of these improvements and in several cases new equipment has been used here on a commercial basis before it has been introduced in the United States.

As far as thorough analysis of results is concerned, it is essential that the reservoir engineer be fully acquainted with the limitations of the basic test data. Further, he must have the ability to interpret these data. With experience, it often happens that seemingly unimportant information may give him a clue to hitherto unexplained reservoir or well behaviour. Needless to say, he is greatly assisted in his work by accurate data and it often happens that he finds that improvements can be made in the test recording and/or reporting which will be of great importance to better evaluation.

Coring and Core Analysis

Of the many different coring techniques, such as conventional, wire-line, sidewall and diamond coring, the last named has furnished the reservoir engineer with the most accurate information, mainly because, with it, high recoveries are obtained. Visual inspection under a microscope provides him with information on the type of pore structure. Gas-oil and oil-water contacts can often be found from a laboratory determination of oil and water saturation of immediately frozen cores. Although this method is suitable for sand and sandstones, it should be borne in mind that freezing of limestone cores can result, later on, in the measurement of permeability being too high, because the expansion of the water upon freezing could cause fractures or fissures between otherwise unconnected pores.

Analysis of permeability and porosity of vugular limestones was improved considerably by the full-size core analysis procedure. Although this procedure is applicable also to sands and sandstones, for these the old-time small plug analysis is still considered satisfactory.

Lately, some doubt has arisen with regard to the accuracy of porosity and permeability results obtained on shaly or silty sands and sandstone. High porosities, and especially high permeabilities, measured in the conventional manner did not correlate with well performance. Could it be that drying of the core plugs after extraction shrinks the shale or silt because interstitial water is driven off? This would create additional pore spaces and passages which would be measured as porosity and permeability. It would be interesting to see some research done on this subject and maybe a method of analysis could be developed in which the shale and silt were left undisturbed for porosity and permeability measurements.

A great many additional measurements can be made on core samples to assist the reservoir engineer in his work, but most of these are still in the research stage. A few are the measurement of connate water content by distillation, restored state method, or capillary pressure curves; relative permeability of oil *versus* gas, and oil *versus* water; flooding efficiency of gas or water; pore size distribution and wettability of the matrix.

Formation Testing

In the language of the oil industry, formation testing is commonly referred to as 'drillstem' testing. Although this term is correct insofar that, generally, the drillstem is used for this type of testing, it

*Trafford & Associates, Petroleum Consultants, Calgary, Alta.

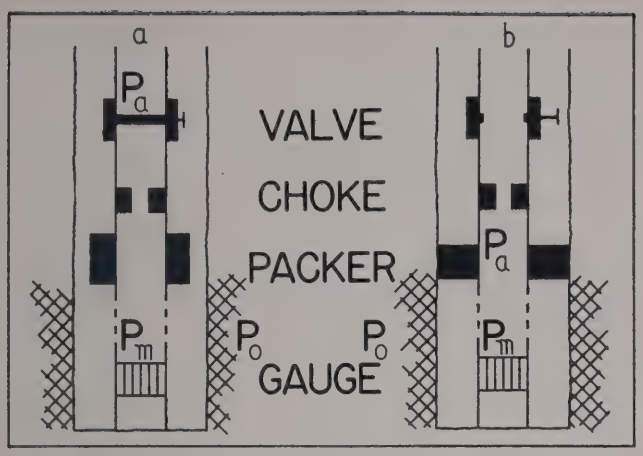

Figure 1.

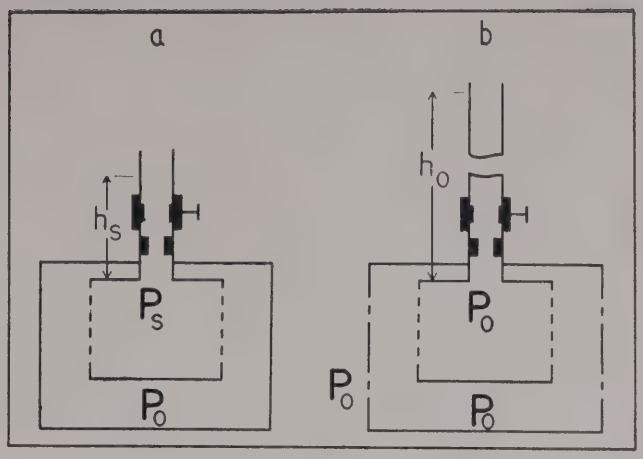

Figure 3.

may be observed that the interest is rather in fluid recoveries than in conditions in the formation (that is, the reservoir) as indicated by the test. Pressure gauges with recording charts have been employed in conjunction with the testers whenever the interval was large enough to accommodate one. In the majority of cases, however, the only use to which the pressure chart was put was to judge whether the tool had opened or not, or whether the packer held. A close study of the pressure charts and of the reservoir mechanism used in conjunction with them reveals immediately their great value to the reservoir engineer.

The author has had the opportunity to study many pressure charts and he has arrived at a tentative formation evaluation factor. This work is still in its initial stages but results obtained so far have been promising. Following is a brief explanation of the method of approach. Figure 1 is a schematic sketch of a tester before setting (1*a*) and an infinitely short time after the trip valve is opened (1*b*). The prevailing pressures at different points are indicated. *Po* is the original oil reservoir pressure, *Pm* is the pressure induced by the weight of the mud column, and *Pa* is, for all practical purposes, atmospheric pressure.

If the formation is dense, no influx of fluids will take place and the recorded pressure remains constant, slightly above atmospheric pressure. If the formation is porous and permeable and holds oil or water, the pressure behaviour can be explained as follows:

The pore spaces immediately connected to the bore hole can be compared to a tank which is pressurized to *Pm* p.s.i.g. when the packer is set and the valve still

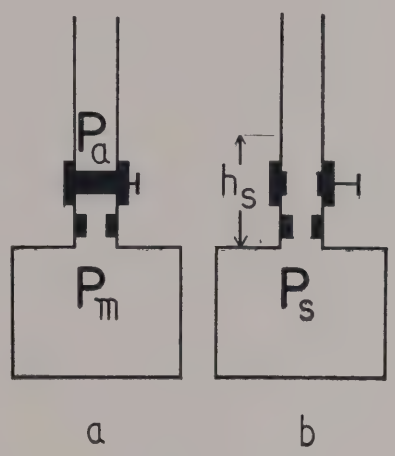

Figure 2.

closed. Figure 2*a* shows this condition. At the moment the trip valve is opened, the pressure tends to fall to *Pa*, but the spontaneous expansion of the liquid in the tank and the lag of the pressure gauge prevents this from being indicated. The fluid will rise inside the standpipe (drill-stem) and a new equilibrium will be reached at pressure *Ps* (Figure 2*b*), the pressure at the end of the surge due to liquid expansion. It can be seen now that a bigger tank would cause a greater surge. Or, reversing this statement, it can be said that, for equal closed-in pressures, a greater surge is an indication of a larger effective tank volume, which is equivalent to a greater porosity and/or better inter-communication of the pores.

The effect of permeability can be explained by imagining the tank of Figure 2 immersed in a larger tank which is still under *Po* p.s.i.g. pressure. This is shown in Figure 3. The walls of the small tank are perforated. Permeability is simulated by the size and density of the perforations. The tank in Figure 3*a* is under the same pressure conditions as that in Figure 2*b*. It can be easily visualized now that liquid will flow from the large tank into the small one and will rise inside the standpipe as a result of pressure differential. In turn, the large tank can be surrounded by a still larger one of *Po* p.s.i.g. pressure and the two tanks again connected through perforations (Figure 3*b*). More fluid will pass through the inside tanks to rise inside the standpipe. Continuing this procedure and allowing enough time to reach equilibrium, the pressure head in the standpipe will eventually equalize with a pressure *Po*, the original reservoir pressure. An increased number of, and/or larger, perforations will tend to speed up the process of equalizing. Or, reversing this statement, the rate of pressure build-up is an indication of permeability. Figure 4 shows this pressure behaviour as it should be recorded on the pressure chart.

At *t*, the tester was started in the hole and final depth was reached at *u* (= *Pm* p.s.i.g.). At *v*, the packer was set and the fluid underneath it was slightly compressed. The trip valve was opened and pressure dropped to *Ps* at *w*. From *w* to *x* the liquid was allowed to enter the tester to complete build-up of *Po* p.s.i.g. Often the tool is closed before equilibrium is reached. If it is allowed to remain closed for 15 to 30 minutes before pulling the packer, the last part of the build-up curve will have the form *x'*. At *y*, the packer is pulled loose and the pressure should record *Pm* p.s.i.g. Between *y* and *z* the tool is withdrawn from the hole.

An accurate pressure chart will show the following:

(*a*) A large surge pressure, *Ps*, indicates good porosity and/or good permeability immediately around the bore hole.

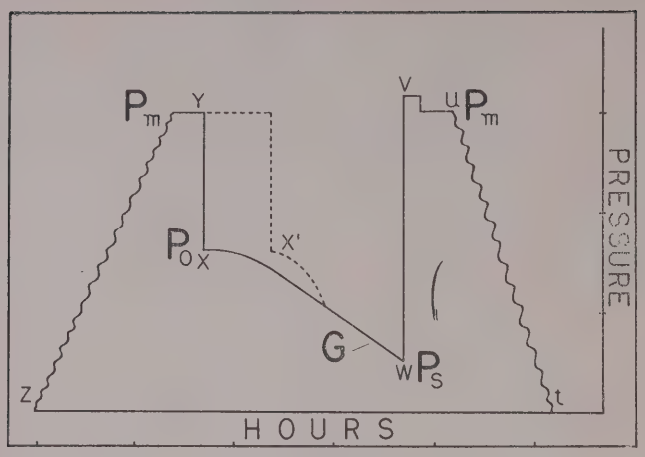

Figure 4.

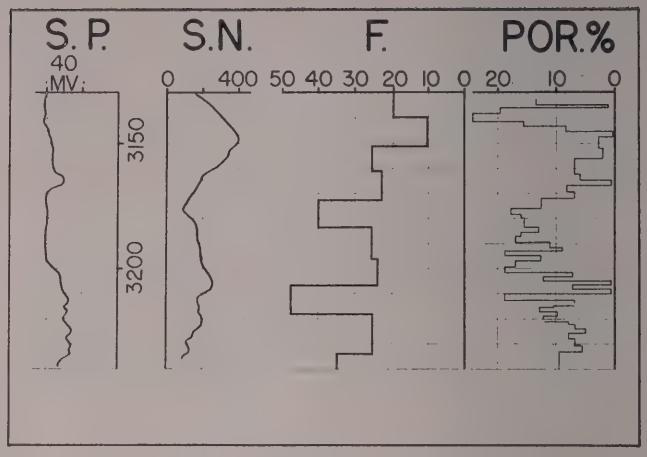

Figure 5.

(*b*) A rapid pressure build-up or steep pressure gradient, *G*, indicates good permeability farther into the formation.

These two factors are closely related. If *Ps* is large, a less steep gradient can be expected because the differential pressure *Po-Ps*, which regulates the flow directly, is smaller. A smaller *Ps* can cause a steeper gradient.

A factor for formation evaluation can be developed by combining the indicative values of porosity and permeability as follows:

$$F = \frac{Po}{Po - Ps} \times G$$

In order to compare formations on the basis of test results when different length intervals and size of drillpipe are used, *Ps* and *G* should be corrected to a common basis. For instance, if the formations were to be compared to a basis of 10-foot intervals, *Ps* and *G* should be multiplied by a fraction 10 ÷ actual interval. If 3½-inch drillpipe is the standard basis for comparison, *G* should be multiplied by a fraction.

$$\frac{(\text{I.D. drillpipe used})^2}{(\text{I.D. } 3\tfrac{1}{2}\text{-in. drillpipe})^2}$$

The effect of bean and hole size has been considered, but is thought too small to affect the result to any appreciable extent. It is the formation permeability which governs the gradient, and the choke permeability is very high compared with this, regardless whether ⅜-inch or ½-inch bean is used. The volume of the hole is a small part of the volume causing a noticeable expansion and may therefore be regarded as identical for holes of different sizes. An example of the correlation of *F* with porosity and the short normal electric log curve is shown in Figure 5.

Thus it appears that, as far as the testing of oilwells is concerned, the importance of accurately calibrated pressure gauges and proper recording mechanisms cannot be over emphasized.

Equal emphasis must be placed on the pressure gauges used when testing gas wells. The formation tester combined with a proper down-the-hole pressure recorder provides a tool to indicate fairly accurately the deliverability of a gas well. Restriction of gas flow at the surface will cause a change in the back pressure on the sand face. Before pulling the packer loose, time should be allowed for a build-up to the original reservoir pressure. A chart as shown in Figure 6 can be expected. Knowing the shut-in pressure, *Po*, and two pressures, *Pf*, while flowing gas at two respective rates, a back pressure curve can be drawn from which the open flow can be derived. Figure 7 illustrates this familiar relationship.

Electric Well Logging

Electric well logging claims the honour of being the oldest means of continuously recording the succession of formations penetrated by the drill bit. Originally used for correlation purposes only, it is now an important tool in estimating rock properties and fluid contents. In addition to the conventional curves, which have progressively improved, many other curves are recorded now, such as the microlog and laterolog.

It is not within the scope of this paper to discuss the mechanism of

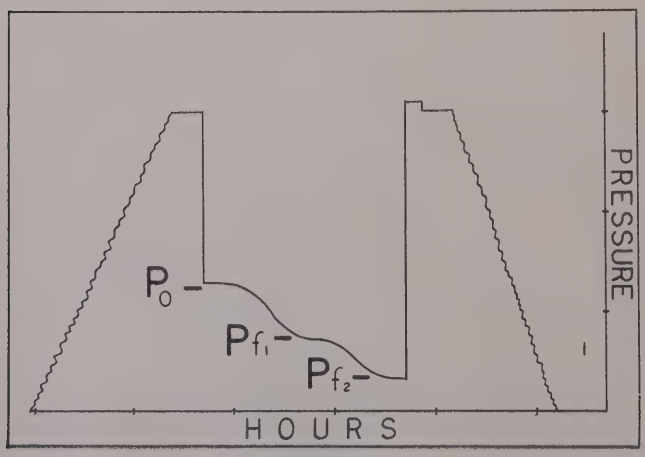

Figure 6.

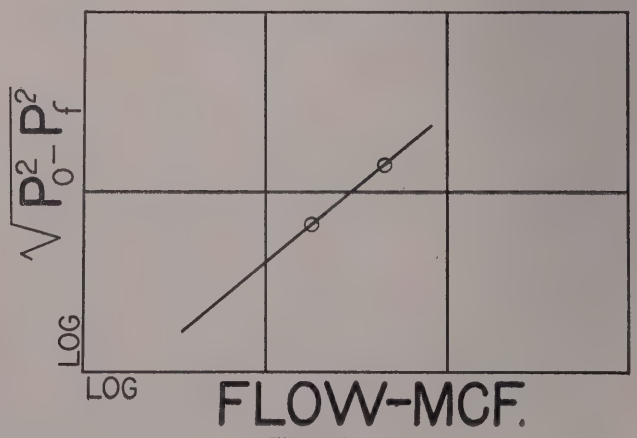

Figure 7.

electric log interpretation. This has been done very capably by many experts. Especially in the limelight recently has been the interpretation of logs in limestone and dolomite formations, and more particularly of logs in the Rocky Mountain region of the United States. To such a degree, in fact, that practically all factors that are used in quantitative evaluation of the Canadian Devonian formations are those derived for the Rocky Mountain region. Factors involved are *n*, Archie's formation exponent in his connate water saturation formula; *m*, which appears in Archie's formation factor formula as an exponent and varies between 1.5 and 2.2; and several others. Small differences in exponential factors have a sizeable effect on log interpretation. It would therefore be of primary importance to establish in the laboratory a value for as many factors as possible of the formations actually encountered in Canadian oil fields.

Radioactive Well Logging

Until very recently the gamma ray curve, in most cases complemented by a neutron curve, has been used mainly as a tool to delineate and distinguish between sands and shales. The neutron curve has been used successfully to detect the presence of dry gas and also to locate the fluid bearing zones in a generally dense limestone section. During the past two years a relationship between porosity and the neutron curve has been developed for limestone formations. Results obtained are very satisfactory and extensive use of the radioactive logging technique has been made in the Redwater field for the purpose of porosity evaluation.

Measuring of Bottom Hole Pressures

In order to follow the behaviour of a reservoir, the reservoir engineer depends to a large extent on the measuring of bottom hole pressures at regular time intervals. The accuracy of the recording pressure gauge is a primary requisite. Also, a proper interpretation of the pressure charts, especially with regard to temperature correction when such is needed, is very important.

Sampling of Reservoir Fluids

The sampling of reservoir fluids under the conditions that prevail in the reservoir, and the subsequent determination and study of the physical characteristics of these fluids, provide the reservoir engineer with basic data for his calculations of reserves on the basis of past performance, and give valuable information on the effect of water-drives and gas caps, the future behaviour of the reservoir, and many other matters pertaining to oil accumulation.

Conclusion

The reader may have received the impression that a reservoir engineer is a collector of all odds and ends within his reach. To a certain extent this is very true. He has never too much material to work with and it is therefore of the utmost importance that the information he obtains from logging and testing be as reliable as possible.

The Heavy Oil Situation in Western Canada

*By J. C. GILBERT**

(*Annual Western Meeting, Calgary, Alta., October,* 1951)

(*Transactions, Volume LV,* 1952, pp. 148-152)

Introduction

THE PURPOSE of this paper is to discuss in a general way the heavy oil situation in Western Canada, and particularly to acquaint those in the oil industry who have not participated in the heavy oil end of the business with past and current developments. The author has covered briefly the location of the heavy and asphalt-base oil fields, their geology, current drilling and production practice, the location of the refineries capable of handling heavy crude, their products and markets, and the problems and future prospects of the industry.

Location and Description of Fields

The term *heavy oil* is generally used to denote crude of 20.9° gravity and under. Since the products from heavy oil are also to a lesser extent obtained from asphalt-base crudes of higher gravity, some mention will be made here of the latter; but in general the emphasis of the paper is on heavy oil as such.

In the order of their production to date, the heavy oil fields and those light oil fields which produce asphalt-base crudes are: Lloydminster, Taber, Vermilion, Conrad, Princess, and Wainwright.

The *Lloydminster* oil field has not only produced most of the heavy oil, but also is currently the largest producer, supplying at the present time about 80 per cent of the heavy and asphalt-base crudes produced in Western Canada. This so-called 'field', 145 miles east of Edmonton, is really a large area about 6 miles wide and 35 miles in length surrounding the town of Lloydminster, on the Alberta-Saskatchewan border. The area, whose initial production was at Dina, in 1929, is divided into a number of sections, or pools, known as: Blackfoot, Durness, Silverdale, Dina, and Kitscoty on the Alberta side; and Silverdale, Lone Rock, Dulwich, Dina-Marsden, and Maidstone-Waseca on the Saskatchewan side. To the end of July, 1951, the area has produced a total of 7,246,739 barrels of oil: 3,158,013 barrels from the Alberta side and 4,088,736 barrels from the Saskatchewan side. Production for 1950 was 1,896,180 barrels. For the first seven months of 1951 it amounted to 1,175,212 barrels, an increase of 11 per cent over the same period in 1950. July, 1951, production was 193,454 barrels, or an average of 6,240 barrels per day, the highest of any month to date. This came from a total of 317 wells, giving for the field an average production of 20 barrels per day per well. The gravity of crude in the area ranges from 9° to 18°.

The *Taber* field, which to date is second to Lloydminster in production of heavy oil, is 125 miles southeast of Calgary and 28 miles east of Lethbridge. It was discovered in 1937 and produces an 18° gravity crude. Cumulative production to August 1st, 1951, amounted to 1,433,073 barrels. Production during 1950 was 114,916 barrels, an average of about 9,576 per month. In July, 1951, it was 17,472 barrels, an average of 564 barrels per day from 18 wells, or 31 barrels per day per well.

The *Vermilion* field, 30 miles west of the town of Lloydminster, is also a heavy gravity (15°) oil field. Discovered in 1939, it had produced 1,248,947 barrels to the end of July, 1951. Production during 1950 totalled 49,041 barrels, or about 4,087 barrels per month. That for July, 1951, was 4,213 barrels, or 136 barrels per day. This came from 11 wells, giving an average of 12 barrels per day per well.

The *Conrad* field, discovered in 1944, is 42 miles east-southeast of Lethbridge. The oil here is a 26° gravity crude of asphalt base. Cumulative production to August 1st, 1951, is 1,096,344 barrels. Production in 1950 amounted to 110,062 barrels, or 9,172 per month. In July, 1951, it was 12,544 barrels, or 405 per day, from 15 wells, an average of 27 barrels per day per well.

The *Princess* field, which first produced in 1939, is 115 miles east and slightly south of Calgary. It is one of the higher gravity (25° to 33°) asphalt- base fields. Production to the end of July, 1951, had totalled 756,862 barrels. During 1950 it was 122,909 barrels (10,242 per month). The output for July, 1951, was 7,027 barrels from 10 wells, an average of 227 barrels per day, or 23 barrels per day per well.

The *Wainwright* field, 40 miles southwest of Lloydminster, is the oldest of the heavy gravity (14° to 20°) oil fields. Production commenced in 1926 and to August, 1951, had totalled 312,835 barrels. Last year's output was 15,360 barrels (1,280 per month) and that for July, 1951, was 1,338 barrels, or 43 barrels per day, from 3 wells, an average of 14 barrels per day per well.

At *Bantry*, not far from the Princess field, some 22° gravity oil was discovered in 1947 and is being produced at the rate of 14 barrels per day. A small amount of asphalt-base oil has been produced at *Barrhead*, about 55 miles northwest of Edmonton, and at *Whitelaw*, some 35 miles west-southwest of Peace River, and negligible amounts at a few other locations.

This year (1951), new sources of heavy crude have been discovered at *Bonnyville*, Alberta, by Tor American, and near *Kindersley*, Saskat-

*Husky Oil Company, Cody, Wyoming, U.S.A.

chewan, 115 miles west-southwest of Saskatoon, by Royalite-Alberean and Husky-Phillips. While these discoveries remain to be evaluated, they do extend the potential black-oil area at Lloydminster 60 miles farther north and 110 miles farther south. In addition, Shell Oil has completed a heavy oil producer on the edge of the *Williston* basin, some 55 miles south-southwest of Regina and 215 miles southeast of Kindersley.

Geology

The heavy oil in the Lloydminster area is found in sandstone formations of Lower Cretaceous age which are believed to be, variously, of marine and non-marine origin. The oil is produced chiefly from the Sparky sand, at depths of 1,800 to 1,900 feet, but small amounts come from the Colony, General Petroleum, and Lloydminster sands. The Sparky sand is the thickest formation and, though lenticular, averages in its two members some 15 to 30 feet; the Colony averages about 10 feet and the General Petroleum and Lloydminster about 15 feet each. The sands are fine-grained, unconsolidated, and even mushy, and consist principally of rounded quartz grains with varying amounts of dark minerals, mainly chert, which gives them at times a pepper and salt appearance.

The oil pools are believed to have resulted primarily from stratigraphic conditions, with structural control playing a secondary part. The sands are thought to have been laid down under deltaic conditions, which would account for their considerable lenticularity. With the regional dip to the southwest, pinching out or shaling out to the northeast favoured the development of stratigraphic traps. Although very little information is available concerning the underlying old Palaeozoic surface in the area, it would appear it was sufficiently uneven and far from flat, as a result of erosion, and perhaps even of structure, that there was a certain amount of draping of the Lower Cretaceous sediments that were subsequently deposited on the uneven surface. It is also possible that some differential compaction took place in these beds. The regional trend of the area is north-northwest; however, the distribution of the pools, though showing general correspondence with this trend, is much less regular.

Although the source of the Lloydminster crude is not definitely known, it is generally considered to be of Devonian origin. It is apparently more than a coincidence that the Lloydminster black-oil belt occurs not far up dip from the eroded portion of the Devonian D*2* and D*3* zones on the east flank of the Alberta syncline.

The above discussion covers Vermilion and Wainwright as well as the pools in the Lloydminster area; however, in those fields the sands are found at slightly greater depths than at Lloydminster, and possibly the structure at Wainwright is more anticlinal in nature. Taber's heavy oil production is from a Lower Cretaceous sand, at a depth of about 3,000 feet. The asphalt-base oil at Conrad is found in a Jurassic sand at about 3,200 feet, and the similar crude at Princess in a Lower Cretaceous sand and in Mississippian limestone, also at about 3,200 feet. At Bantry, production is from the Lower Cretaceous; at Barrhead, Mississippian; at Whitelaw, Permo-Pennsylvanian; and at Shell's discovery in the Williston basin, Mississippian dolomite.

It is interesting to note that, whereas most of the light oil in Western Canada is produced from the Devonian, most of the heavy crude is found in the Lower Cretaceous.

Drilling and Production Practice

The preferred method of searching for black oil appears to be, first, to locate a structurally 'high' area in the Lower Cretaceous by means of surface geology mapping, seismic surveys, or core holes, and then, by a systematic drilling programme, to locate the possible oil pool on the 'high'.

Drilling the average 1,900-foot Lloydminster well presents no particular problems. Light, portable rotary drilling rigs are used and a well can be completed — that is, spudded, surface casing set, hole drilled to total depth, production casing cemented, and the rig moved off — in four to six days.

On wildcat locations, the first well in an area is usually cored, electric logged, and timed for drilling rates. Coring is generally omitted on subsequent wells, correlation

Production from Heavy and Asphalt-Base Oil Fields in Western Canada
(Barrels)

Year	Lloyd-minster*	Taber	Vermilion	Conrad	Princess	Wainwright	Others	Total
Prior to 1937	15,073(a)					99,425(b)		114,498
1937	1,642	600				13,459		15,701
1938	6,383	15,098				12,985		34,466
1939	3,981	3,720	202		515	11,624		20,042
1940	6,725		10,817			7,527		25,069
1941	3,310	5,600	22,051		19,587	11,733		62,281
1942	3,197	29,819	56,819		10,478	14,510		114,823
1943	2,840	88,735	93,258		340	18,136	462	203,711
1944	6,296	148,638	234,603	24,733	13,815	17,154		445,239
1945	44,829	135,000	238,358	143,696	63,377	16,472		641,732
1946	213,061	206,925	183,946	212,645	64,953	15,114	1,558	898,202
1947	848,511	205,236	138,401	202,929	106,920	18,325	145	1,520,467
1948	1,509,058	201,527	112,331	179,627	186,393	17,131	16,519	2,222,586
1949	1,510,451	150,746	86,933	139,728	121,227	16,086	18,202	2,043,373
1950	1,896,180	114,916	49,041	110,062	122,909	15,360	19,954	2,328,422
1951 (7 months)	1,175,212	126,513	22,187	82,924	46,348	7,794	5,080	1,466,058
Total	7,246,749	1,433,073	1,248,947	1,096,344	756,862	312,835	61,920	12,156,730

*Comprising the Blackfoot, Durness, Silverdale, Kitscoty, and Dina sections in Alberta, and Silverdale, Lone Rock, Dina, Dulwich, McLaren, Waseca, and Maidstone sections in Saskatchewan.
(a) Years 1929 to 1936, inclusive.
(b) Years 1926; 36, inclusive.

being made by means of electric logs and drilling time charts, with occasional checks when necessary with sidewall samples. In general, drill stem testing for heavy crude at Lloydminster is not satisfactory, as frequently good sands will give negative tests, no fluid, or very little, entering the drill stem.

The general practice is to set 10¾-inch surface casing at 150 or 200 feet and drill to total depth, setting 7-inch casing through the oil sand. Although some 5½-inch casing is used, 7-inch is more satisfactory from the standpoint of pumping and working on the well. Cement jobs on the production casing have given trouble frequently in the past. This difficulty has been largely eliminated by more care of the drilling mud, particularly at the time of cementing, together with the use of centralizers on the casing.

After the drilling rig is moved off the well, a servicing rig is moved on to perforate the casing and place the well on production. This job usually consists of bailing the mud and water from the hole, allowing to stand a few hours to test the casing, filling the casing with water, perforating, bailing to remove most of the water and whatever sand comes in, and running of the pump, tubing, and rods. An average job requires three to four servicing days. Occasionally, more time is required when considerable sand comes into the well and it is necessary to run tubing to circulate the sand out, or when the oil will not readily flow into the well, and hot water, Diesel oil, and other treatments are required. Both gun and jet perforators are used, the guns being favoured for the lighter gravity wells and the jet for the lower gravity, more viscous oil, wells.

Generally, 3-inch upset tubing is used, but 2½-inch upset is satisfactory if the oil is not too heavy. Both tubing liner and insert pumps are used, the insert type being the better suited to the higher gravity, less viscous crudes. Gas anchors are used and pumps are usually set about 150 to 200 feet off bottom, and, after six months to a year, are lowered to 50 feet or so off bottom. Wells are powered usually by gas engines, but, where available, electric power is frequently used because of its convenience.

The highly viscous nature of the crude together with its sand content, especially under the low winter temperatures prevailing at Lloydminster, make it impractical (except in a few cases) to use central batteries or to move the oil by pipeline to the refineries. Instead, it is hauled by tank trucks. In order to load the trucks by gravity, the practice has been to erect a 250- or 500-barrel tank on a wooden platform to give the necessary height. Newer practice is to have a three-ring 750-barrel tank, the bottom ring taking the place of the platform and acting as space for the accumulation of B. S. & W. and for the draining of water. The oil is gravitated from a connection just above the bottom of the first ring. Each tank has a heating arrangement for keeping the oil warm.

Various means have been tried to keep the sand out of the wells, but, because of the high viscosity of the oil and the fineness of the sand, no screen or other method has been found which will keep all of the sand out and still let the oil come in. Perhaps the most effective approach has been through control of fluid levels and production rates.

The average initial production of wells in the Lloydminster area is small, probably in the range from 25 to 50 barrels per day. In certain areas such as Lone Rock, Silverdale, and Blackfoot, wells have produced initially as much as 100 barrels and more per day. The tendency of such wells is to settle down rather rapidly to 30 to 40 barrels per day and then to decline more gradually. For all practical purposes, there has been, from year to year, no curtailment of oil production in the heavy oil and asphalt-base fields. It is noteworthy that all the wells have a relatively low production rate, from 31 barrels per day per well at Taber down to 12 barrels per day per well at Vermilion. At Lloydminster the average production during July, 1951, was 20 barrels per day per well with a range for the various pools of 7 barrels to 28 barrels per day per well. There is no set spacing pattern at Lloydminster, as wells are drilled on both the 10-acre and the 20-acre patterns; however, the 20-acre pattern appears preferable.

At the present time (August, 1951), ten rigs are operating in the Lloydminster area. Seventy-one producing oil wells have been completed during the year, which, with the additional 50 to 60 planned for this fall, will make about 130 for 1951, compared to 99 for 1950.

PRODUCTION DATA BY SECTIONS, LLOYDMINSTER AREA
(Barrels)

PROVINCE AND SECTION	PRODUCTION TO END JULY, 1951	PRODUCTION JULY, 1951	NO. OF WELLS ON PRODUCTION JULY, 1951	DAILY AVERAGE PRODUCTION PER WELL, JULY, 1951
ALBERTA:				
Blackfoot	1,896,284	35,412	59	19
Durness	778,495	23,412	44	17
Silverdale	346,001	5,882	17	11
Dina	82,934	567	1	18
Kitscoty	54,299	7,253	12	20
TOTAL	3,158,013	72,526	133	18
SASKATCHEWAN:				
Silverdale	2,111,353	34,440	68	16
Lone Rock	1,675,395	67,787	79	28
Dulwich	213,613	12,082	19	20
Dina-Marsden	66,512	5,957	15	13
Maidstone-Waseca	21,863	662	3	7
TOTAL	4,088,736	120,928	184	21
TOTAL, ALTA. AND SASK.	7,246,749	193,454	317	20

REFINERIES

The refineries in Western Canada with facilities for handling heavy crude can be divided into three classes: those which operate on heavy crude and manufacture refined asphalt products; those which dehydrate and de-salt heavy crude; and those which operate primarily on light crude but do process some crudes for their asphalt content.

In Alberta, Supreme Refineries have a 500-barrel per day asphalt plant at Ogden, and Excelsior Refineries operate a 2,500-barrel per day plant and Husky Oil & Refining a 7,500-barrel per day plant at Lloydminster; while in Saskatchewan, Moose Jaw Refineries operate a 1,400-barrel per day plant at Moose Jaw and Prince Albert Re-

fineries a 1,000-barrel per day plant at Prince Albert.

The dehydration and de-salting facilities are at Borradaile, Alberta, where Husky Oil and Refining operate a dehydration plant with a capacity of 1,200 barrels per day for the processing of bunker fuel oil, and at Lone Rock, Saskatchewan, where Franco Oils operate a dehydration and de-salting plant with a capacity of 2,000 barrels per day.

Operating primarily on light crude but equipped also to produce asphalt products are the British-American refineries in Calgary and Moose Jaw and those of Imperial Oil in Calgary and Regina. It may be mentioned that Imperial, Shell, and Standard Oil of B.C. have plants of this type in British Columbia, but they do not operate on Western Canada crude.

For the sake of brevity, comments on refining will be restricted to the plants operating on Lloydminster crude. In general, the handling and processing of this heavy crude is more difficult than with light crudes because of the low gravities and high viscosities, particularly under the low temperature conditions prevailing during winter.

The most unusual phase of the operation is the transportation and handling. As mentioned previously, it is not practical to transport the crude from the fields to the refineries by pipeline. It is therefore hauled to the plant by tank trucks with capacities of 40 to 80 barrels each. Since the producer does not dehydrate the oil (except Franco Oils, at Lone Rock), the oil, sand, and water which cannot be drained off at the well are all hauled to the plant. The average water content of this wet crude is about 20 per cent, and the sand content about 0.4 per cent. The wet crude is weighed when it reaches the plant, and the weight is converted into barrels, which is the basis for payment, using a table of factors which vary with the gravity. The crude is measured by weight rather than by volume because of the tendency of the heavy oil to retain gas. After weighing and sampling, the wet crude is dumped into a below-ground storage tank equipped with steam heating coils. Much of the sand settles out in this tank, and is periodically removed.

The wet crude is then pumped through dehydrators, which remove the water and most of the remaining sand. Due to the highly abrasive nature of the sand, there is considerable wear on the charging pumps to the treaters, and on certain parts of the treaters themselves. After dehydration, the crude is de-salted. Its salt content is about 4,500 pounds per 1,000 barrels and about 98 per cent of this is removed. The dehydrated and de-salted crude is then processed by conventional topping or fractionation to produce the various refined products.

Western Canada Refineries with Facilities for Handling Heavy and Asphalt-Base Crudes

Province and Company	Location	Crude Charigng Capacity
Alberta:		
British American Oil Co., Ltd.	Calgary	(a) 7,200 brl/day
Excelsior Refineries	Lloydminster	2,500 "
Husky Oil & Refining, Ltd.	Borradaile	(b) 1,200 "
Husky Oil & Refining, Ltd.	Lloydminster	7,500 "
Imperial Oil, Ltd.	Calgary	(a) 9,200 "
Supreme Refineries, Ltd.	Ogden	500 "
British Columbia:		
Imperial Oil Ltd.	Ioco	(a) 12,000* "
Shell Oil Company	Shellburn	(a) 8,500* "
Standard Oil Co. of B.C.	Burnaby	(a) 8,350* "
Saskatchewan:		
British American Oil Co., Ltd.	Moose Jaw	(a) 6,000 "
Franco Oils, Ltd.	Lone Rock	(c) 2,000 "
Hi Way Refineries, Ltd.	Saskatoon	(a) 3,150 "
Imperial Oil, Ltd.	Regina	(a) 20,000 "
Moose Jaw Refineries	Moose Jaw	1,400 "
Prince Albert Refineries	Prince Albert	1,000 "

(a) Includes skimming and cracking capacity.
(b) Dehydration only.
(c) Dehydration and de-salting.
* These refineries do not operate on Western Canada crude.

Products and Markets

The types of product and the proportions of each produced from Lloydminster crude depend on the character of the crude itself and on the demand for the several products at any particular time. Although the demand is quite variable, especially for asphalt products, on the average about 20 per cent of the crude is refined into light products, such as gasoline, burner fuels, and Diesel fuels; 40 per cent into finished asphalt products, such as paving, industrial, building, and briquetting asphalts; and the balance, or 40 per cent, is processed into bunker fuel oil for railway and industrial use. Lloydminster crude, when refined, yields not only a high quality asphalt but also a gasoline of better than average antiknock characteristics and a domestic fuel with an unusually low pour point (−70°F.).

The largest market for finished asphalt in Western Canada is, of course, that for road building. This year, the weather slowed down that part of the business to a certain extent; however, the defence programme has stepped-up the demand for asphalt for airports. The development of the industrial and building asphalt field has resulted in two plants being erected close to sources of supply. At Lloydminster, Sidney Roofing has made a major investment in a plant adjacent to a refinery, from which it takes its raw material direct in a hot stream. At Edmonton, Building Products have also erected a plant near the refineries in that city. Asphalt for briquetting coal is now an important part of the business and provides a year-round market. Bunker fuel oil has been, and still is, the largest volume product.

Asphalt products from Lloydminster crude now are being marketed not only in Alberta and Saskatchewan but also in British Columbia, Manitoba, and Ontario. During 1950, Alberta and Saskatchewan refineries shipped 710,090 barrels of asphalt products, of which 64,982 barrels went to British Columbia, 64,649 to Manitoba, and 32,272 to Ontario.

Problems and Prospects

With the development of heavy crude production over so large an area at Lloydminster and the discovery of heavy crude as far north as Bonnyville, Alberta, and as far south as Kindersley, Saskatchewan, with production as far west as Wainwright, Alberta, and showings as far east as North Battleford, Saskatchewan, it appears that there is an immense area, over 200 miles long and as much as 100 miles wide, favour-

able for the discovery and development of sufficient heavy oil production to meet the requirements of heavy crude in Western Canada for some time to come. A systematic exploration programme is required for the continued discovery of new pools, and, once the pools are discovered, adequate development programmes must be instituted and carried on to bring the production up to the amounts required, in view of the average low production per well.

In the past, on account of the shallow depths and relatively low drilling costs, some inexperienced operators have participated in the development of the Lloydminster area. This type of operator is on the decline. With this decline it is believed operations should be generally more successful, fewer dry holes should result, water problems should be lessened, and more wells should be carried on to successful completion and production. This will tend to reduce the cost of development and increase the return for the field as a whole. With drilling and production in the hands of more responsible operators, field water problems, gas control problems, etc., will receive more of the attention due them.

Experiments are now being carried on with the object of finding or developing a satisfactory, inexpensive, well-type dehydrator. Success in this endeavour will reduce transportation costs by reducing the amount of water hauled. The trend toward larger capacity trucks is also working in the same direction. In addition, grading, graveling, and oiling of roads will aid in reduction of these charges and will tend to even out the delivery of oil from the wells to the refineries.

In the refining end, the trend is toward more and more finished products as markets are developed. In Canada, particularly Western Canada, there are thousands of miles of roads which await paving or some type of improvement which would utilize asphalt. This road work and the development of the country undoubtedly will go hand in hand. The field of specialty products, such as roofing asphalt, briquetting asphalt, and the various industrial and building asphalts, is also bound to expand. Western Canada has large coal reserves which can be made more marketable through processes such as briquetting. Research is continually developing new products and new markets. While bunker fuel oil has represented the most important part of the market for heavy crude, it is believed that, in five to ten years if not sooner, the market for this product will decline. The dieselization of Canadian railways is only a matter of time and natural gas is bound to take the place of fuel oil for industrial purposes when it is made available. This trend, however, will have its compensations by leaving more of the crude available for the finished product market.

Additional markets for heavy crude in Western Canada, both east and west, await the discovery of new sources of supply and the construction of pipeline facilities for the handling of blended heavy crude. When it is considered that shipments of asphalt from Canadian refineries amounted to 4,357,974 barrels in 1950, and that Western Canada supplied only 710,090 barrels, or 16.3 per cent, of this total — the balance being produced from imported crude — it is apparent that, given the means of economically getting the product to market, either through cheaper transportation or an exchange or trading arrangement with the United States, a great potential market exists for Western Canada heavy crude which is now supplied by imports. When the mechanics are worked out to meet this demand, a tremendous impetus will be given to the heavy and asphalt-base oil development of Western Canada.

Acknowledgments

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Operation of the Creighton Mill

Of The International Nickel Company of Canada, Limited

By THE MILL STAFF

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Introduction

COMMENCEMENT of operations at the new No. 7 shaft and 10,000-ton mill at Creighton mine was a major step in the programme of conversion to all-underground mining by the International Nickel Company of Canada, Limited. Planned ten years ago, the orderly transition has been greatly speeded-up to offset the heavy drain on production imposed by World War II and also to meet the increased demand for nickel caused by the current international situation. Development of new mining methods, and process changes in the reduction plants, have made possible the economic recovery and utilization of ore of lower grade than formerly used and of varying metallurgical characteristics. By 1953, when the Company's open-pit mining operations are scheduled for completion, the hoisting capacity of its five underground mines in the Sudbury district will have been increased to 13,000,000 tons per year, compared with 5,700,000 tons of underground ore hoisted in 1950.

Two types of ore occur at Creighton. One is a breccia and stringer type that contains a high percentage of massive sulphides and is located in sheets along shears and in large and small pockets on the underside of the main norite mass; the other type, forming the main part of the ore zone, is mineralized norite that occurs as a sheet up to 400 feet thick overlying the breccia ore. In both types of ore the minerals of economic interest are pentlandite, which is intimately associated with pyrrhotite, and chalcopyrite. Early mining was confined to the breccia ore; this ore was mined by open stoping methods. The hanging-wall was later allowed to cave, with the result that the old stopes are now filled with caved low-grade ore.

Exhaustive research showed that recovery of the low-grade hanging-wall ore was economically feasible by using the caving method of mining, and by handling large tonnages and concentrating on the spot instead of transporting the ore to the Copper Cliff mill.

Laboratory and 10-ton pilot plant test work, followed by an operational-scale test in the Copper Cliff mill, preceded the designing of the new Creighton mill. The operational-scale test was on a 1,500-ton-per-day milling basis over a three-month period, and information gained on grinding and flotation practice was used for laying out a Creighton mill flow-sheet.

The Copper Cliff mill tailing disposal area, located four miles from Creighton, was the logical area to use for disposal of tailing from the new mill. Since this would necessitate the building of new trestle and pumping stations to connect the Copper Cliff and Creighton mill tailing disposal systems, the feasibility of pumping bulk concentrate from Creighton to the Copper Cliff mill was investigated. A pumping installation and 4,000 feet of wood stave line was installed at Copper Cliff, and concentrates of varying pulp densities over a wide tonnage range were pumped through this line twenty-four hours a day for a three-week period. Information obtained on pump capacity, head loss in the line due to friction, and maximum pulp velocities allowable, showed that concentrate could be pumped from Creighton to Copper Cliff economically by utilizing the personnel at the tailing pumping stations required for the Creighton mill plus the men in the existing stations in the Copper Cliff system, and enlarging these stations to handle a second product.

Therefore, building a mill at Creighton would provide low-cost ore processing for the following reasons: elimination of ore transportation costs; low-cost transportation of concentrate *via* pipe line to Copper Cliff through the interlocking of the two tailing disposal systems; provision of a source of mine fill from mill tailing which could be sent direct to the underground workings.

The Creighton mill is built at the site of Creighton mine No. 7 shaft, approximately six miles west of the International Nickel Company's reduction plants at Copper Cliff. The No. 7 shaft headframe and hoisthouse are integral parts of the mill building and ore is hoisted direct to the crushing plant coarse-ore bin. As originally designed in 1948, the mill was to treat 6,000 tons of ore per day from the deposit of low grade ore mentioned previously. Due to the growing unrest in world affairs and the increased demand for nickel, plans were altered during construction to increase the mill capacity to 10,000 tons per day. This increase provided for the treatment of 2,000 tons per day of ore from Creighton No. 5 shaft, located two-thirds of a mile distant. The No. 5 shaft ore, from which the magnetic portion is removed for direct smelting before transferring by conveyor to the mill, is of higher grade than the No. 7 shaft ore and is kept separate throughout the milling process.

Preliminary operation commenced on February 1st, 1951. Three-shift operation commenced on February 20th, handling 2,000 tons per day of No. 5 shaft ore until July 24th, when No. 7 shaft began operating. During this low-tonnage period, all equipment in the mill was operated at different intervals in preparation for the higher tonnage rate. By the end of August, No. 7 shaft was in full production and the mill has been operating at rated capacity since that date.

Figure 1.—Creighton mill.
Tailing pipes in foreground.

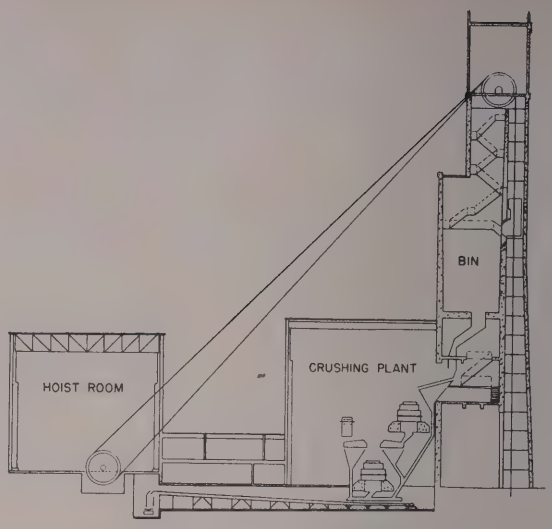


Figure 2.—Section through headframe and crushing plant, showing one of the parallel crushing circuits.

The mill building is a modern fireproof structure of steel, concrete, and tile covering an area 440 ft. by 175 ft., and is 70 ft. high. It is built on level ground and laid out in three parallel sections: the bin section, 440 ft. by 54 ft., containing change house, offices, and grinding steel storage bays, in addition to the 20,000-ton ore bin; the crushing and grinding aisle, 440 ft. by 60 ft., including a 100-ft. repair bay; the flotation aisle, 320 ft. by 60 ft., also containing reagent storage rooms and the classifiers of the sand fill plant. Beyond the sand plant, immediately adjoining the crushing plant, is the reinforced concrete headframe of No. 7 shaft, towering 196 feet high. The crushing and grinding aisle is serviced by a 35-ton cab-operated overhead travelling crane with a 10-ton auxiliary hook, and the flotation aisle by a 5-ton overhead travelling crane operated from the cell floor. A 10-ton electric cable hoist services the rod storage section. The plant offers ideal working conditions; the building is well lighted with incandescent, mercury vapour, and fluorescent lights, and adequate ventilation is provided by fans housed in the roof.

Crushing

After receiving primary crushing underground in Traylor 48-in. by 66-in. crushers set at 6-in. opening, ore is hoisted in No. 7 shaft at a maximum rate of 700 tons per hour in 15-ton skips which dump directly into a 700-ton bin. This bin forms part of the headframe and is of reinforced concrete construction lined with 2-in. Ni-Cr-Moly plate.

The feed flows by gravity through the crushing plant; the top of the coarse-ore bin is at an elevation of 105 feet above the conveyors handling the crushed product. A cross-section of the plant, which is located at the end of the grinding aisle adjoining No. 7 shaft headframe, would show, in descending order of elevation: a Ross feeder, a Tyler Ty-Rock screen, a Symons Standard crusher, two Symons Rod-Deck screens, a Symons Short Head crusher, and a conveyor handling the crushed product. There are two complete crusher circuits, operating in parallel.

The Ross feeder, situated under the coarse-ore bin, has a feeding capacity of from 200 to 600 tons per hour and is driven by a d.c. motor. It discharges to a 6 ft. by 14 ft. Tyler Ty-Rock scalping screen equipped with a ½ in. by 2 in. opening screen cloth protected by a grizzly deck with 3 in. by 3 in. openings. The screen oversize feeds direct to a 7-ft. heavy duty Symons Standard crusher equipped with a coarse bowl and set at 1¼-in. opening. The Standard crusher product drops to a 5 ft. by 8 ft. Symons Rod-Deck screen with 7/16-in. clear opening between rods. The screen oversize discharges to a 7-ft. heavy-duty Symons Short Head crusher equipped with a fine bowl and set at ¼-in. opening. Screen undersize drops directly to the conveyor below the Short Head crusher through gravity chutes.

The plant is operated from a central location over the Short Head crushers. The Ross feeder drive motor speed is controlled by a rheostat located in this area, and ammeters on crusher motor circuits are also mounted in this location. The feed rate to the crushers is adjusted to maintain maximum power input to the Short Head crusher motors. The Short Head crushers are set daily to maintain a uniform product for rod-mill feed.

All crushers are driven by 300 h.p., 440 r.p.m. motors, direct connected. The motors are cooled by air forced through a duct system and exhausting from the motors into the building. Each crusher has its own pressure oil system and oil settling tank. Mercoid pressure switches installed in the oil supply lines automatically kick out the crusher motors if the oil pressure fails. Recording thermometers installed on the crusher return oil lines sound an alarm if the oil temperature rises above the normal operating range. All crushers are equipped with water seals.

Ore from No. 5 shaft, comprising 20 per cent of the total mill tonnage, is crushed in a 7-ft. Symons Standard crusher, with 2½-in. setting, at No. 5 shaft rock-house, and is separated into two products with a magnetic head pulley. The non-magnetic portion of the ore is stored in a 2,500-ton bin, from which it is transported to the mill by a semi-enclosed system of three main conveyors in series. An open walkway extends the full length of the conveyor system. The walkway side of the conveyor housing is open but, despite this, the system has operated with no difficulty in temperatures of -40°F. The 30-in., 6-ply, 42-oz. belts travel at 400 feet per minute and have a rated capacity of 600 tons per hour. The second of the two transfer points in the

conveyor system contains a 150-ton surge bin equipped with variable-speed roll feeders with remote controls at the mill to maintain the desired rate of feed to the receiving belt ahead of the crushers. The conveyor enters the crushing plant at an elevation above the Short Head crushers. The ore discharges from the receiving belt to two 5 ft. by 8 ft. Symons Rod-Deck screens with 7/16-in. clear openings between rods. The screen oversize discharges to the No. 7 shaft Short Head crushers and the crusher product and screen undersize drop to the conveyors located below the crushers. The No. 5 shaft ore is crushed during the period No. 7 shaft is inoperative for shaft inspection and repair.

The two parallel conveyors, located under the crushers, discharge onto a single 42-in. conveyor. This conveyor is separately housed and runs up the south side of the building, a distance of 460 feet, on a 15-degree incline. A Merrick weightometer installed on this belt records the total tonnage crushed. A Vezin automatic sampler samples the ore as it discharges onto a transfer conveyor, which delivers the ore to the final conveyor running the full length above the 20,000-ton fine-ore bin. This bin is divided into four 5,000-ton sections, three reserved for No. 7 shaft ore and the fourth for No. 5 shaft ore. A tripper equipped with a shuttle conveyor, installed on this conveyor, distributes the ore to any one of four parallel openings the full length of the bin. Distributing the feed in this manner prevents any segregation of coarse and fines in the bin.

Crusher wearing parts changed to date have been insufficient to provide figures on crusher steel consumption. Screen analyses of crushed product, and power consumption, are shown in Tables I and II.

Dust Control

Dust from the screens and crushers is piped by a system of ducts to a collecting unit which consists of 20 standard 34-in. conical wet dust-collecting cyclones through which air is drawn at the rate of 62,000 cu. ft. per minute at 5-in. static pressure. The dust is withdrawn from the bottom of the cyclones and pumped to the flotation circuit in the form of a sludge.

Grinding

The grinding section consists of eight 10 ft. 8 in. by 13 ft. Marcy trunnion-type overflow mills and four 12-ft. quadruplex Dorr *F.X.* classifiers. The grinding section is divided into four units, each containing a rod mill, a ball mill, and a classifier. Ore to the unit is fed direct to the rod mill, and the rod mill discharge is pumped to the classifier. The classifier rake sands feed the ball mill, which operates in closed circuit with the classifier.

All mills are driven by 800 h.p., 225 r.p.m. synchronous motors direct connected to the mill pinion shaft. The mill drive is through double helical gears. The motors are operated by push-button control at the mills; automatic starter control equipment is located in switch rooms. The rod mills operate at 59.5 per cent of critical speed, the ball mills at 74.2 per cent. Mill speeds may be varied over a wide range by changing the mill pinions. The rod-mill discharge trunnion liners have a clear opening of 31-in. and a trommel screen of ½-in. punched plate is attached to the trunnion liners to prevent broken rod steel from entering the mill discharge sump. The ball-mill discharge trunnion liners have a clear opening of 60-in., and a ball retaining grate of nickel cast-iron with ¼-in. slotted openings is installed in the mill discharge. Two Bowser oil systems circulate oil to mill trunnion and pinion bearings. Each oil system services four mills. A Mercoid pressure switch is installed on the oil feed line to each mill and automatically trips the mill motor if the oil pressure drops.

The mills are charged with 70 tons of grinding media: 3-in. and 3½-in. rods in the rod mill, and 1½-in. chill cast nickel alloy balls in the ball mill. Some 1½-in. and 2-in. forged steel balls were included in the original ball charge. The original steel charges have not been completely replaced, so steel consumption rates cannot be considered representative. Rod steel consumption to date is 1.12 lb./ton, and ball steel 0.71 lb./ton, of ore ground. Steel is added to the mills three times weekly, the amount depending on the daily mill motor power readings. Rods are charged to the mills by a power-driven machine.

All mill liners are of chill cast Ni-Hard alloy with the exception of the feed end rod mill liners, which are of manganese steel. Two types of shell liners are in use; one a flat liner 4¼ in. thick, the other a cast lifter-type liner of 3-in. minimum thickness and 5-in. thickness through

Table I

Mesh	Crushing Plant Product — Rod Mill Feed			
	No. 5 Shaft Ore		No. 7 Shaft Ore	
	Percent by Weight		Percent by Weight	
	This mesh	Cumulative	This mesh	Cumulative
1.050 in			4.1	4.1
0.742 in			8.0	12.1
0.525 in	19.7	19.7	5.8	17.9
3	32.3	52.0	29.6	47.5
4	9.9	61.9	7.1	54.6
8	9.3	71.2	10.5	65.1
10	3.4	74.6	4.4	69.5
14	3.2	77.8	3.7	73.2
28	4.7	82.5	5.3	78.5
35	2.2	84.7	2.4	80.9
48	1.8	86.5	2.2	83.1
65	1.9	88.4	1.8	84.9
100	1.9	90.3	1.3	86.2
150	1.8	92.1	2.7	88.9
200	1.0	93.1	1.2	90.1
—200	6.9	100.0	9.9	100.0

Table II

	Tons Original Feed* Per Crusher Hour	Power Consumption K.W.H./ Ton
Standard Symons	400	0.38
Short Head Symons	400	0.51

*Total tons to the plant pro-rated on the crusher operating hours, regardless of removal of intermediate fines.

the lifter section. Test work on the 6½ ft. by 15 ft. mills at Copper Cliff, operating at 76 and 81 per cent of critical speed, showed no advantage when using the wave-type liners instead of the present flat liners for either rod or ball service. In order to test the two types of liners in the slower-speed large-diameter mills at Creighton, two mill units were lined with the lifter-type liner. In one of the other sections, the ball mill was installed with flat liners, and in the remaining unit the rod mill was equipped the same way; the other mill in both units has the more conventional wave-type liner. Under present operating conditions there is no indication that either type of liner is superior from a tonnage standpoint. Since there have been no liner changes required to date, comparative figures are not available for liner steel consumption.

As mentioned earlier, the 20,000-ton fine-ore bin is divided into four 5,000-ton sections, one for each grinding unit. Ore is drawn from each 5,000-ton section by 16 variable-speed roll feeders, mounted in four banks of four, and discharging to a series of conveyors which collect and transfer the ore to the rod mill feed conveyor. This conveyor is equipped with a Toledo Chronoflo scale which records the tonnage and automatically maintains any set feed rate by regulating the speed of the roll feeders. The mill feed conveyor discharges the ore into the rod mill feed box, where water is added to give 70 to 72 per cent solids in the mill discharge. The feed enters the mills through a pipe inserted in the feed trunnion. The discharge of both the rod and ball mill is laundered to a sump, centrally located between the mills and equipped with two 6-in. Wilfley pumps using Ni-Hard wearing parts. The pumps deliver to the classifier situated above and to one side of the ball mill, and the classifier rake sands are laundered to the ball mill. Water is added to the rake sands to maintain 70 per cent solids in the ball mill discharge, and the classifier overflow is controlled at 39 per cent solids. Continuous-recording flow meters are installed on the water supply of each unit, allowing an accurate control of grinding circuit pulp densities and eliminating the necessity of an automatic density controller. Each grinding unit has a clean-up bucket elevator, centrally located, to provide a quick and easy method of cleaning-up spills. The three grinding units operating on No. 7 shaft ore yield a classifier overflow product containing 13 per cent +65 mesh material at a feed rate of 100 tons per hour per unit. The fourth unit, operating on No. 5 shaft ore, will give a comparable product with a feed rate of 115 to 120 tons per hour. The three classifiers of the No. 7 shaft ore grinding units overflow to a common sump ahead of flotation. The overflow of the fourth classifier, operating on No. 5 shaft ore, is laundered to a separate sump but may be diverted to the No. 7 shaft ore sump.

Figure 3.—Grinding aisle.

Screen analyses of the grinding circuit products are given in Table III:

Flotation

Flotation practice consists of a straight recovery circuit from which a bulk copper-nickel concentrate is produced. The wide difference in grade of ore from the No. 5 and No. 7 shafts necessitates two separate circuits, identical in all respects with the exception of the number of flotation machines used in each. Both circuits may be combined to form a single circuit utilizing all flotation cells and will be treated as such for ease of description. Ratio of concentration averages 5.6 to 1.

Flotation feed is pumped to a six-way distributor and flows by gravity to six parallel rows of No. 30 Denver cells. There is a total of 144 cells, 24 to each row, arranged in three banks of eight cells in series. The cells are equipped with standard rubber-covered receded disc impellers operating at 266 r.p.m. and no auxiliary air is used. One 25 h.p. motor operates two cells. The first twenty-one cells of each bank produce the bulk copper-nickel concentrate; the concentrate from the last three cells flows to a central sump and is re-distributed to the ninth cell of each bank as a control circulating load. The bulk concentrate is laundered to a sump from which it is pumped a pipe-line distance of 7½ miles to the Copper Cliff concentrator, where it is separated into

its final products of nickel and copper concentrate.

Flotation reagents are added to the feed and circulating sumps and are also step-fed to the cells. Lime is used to maintain the proper pulp alkalinity, potassium pentasol xanthate is the collector, copper sulphate the activator, and pine oil the frother. Sodium silicate is used for pulp dispersion. All reagents are fed at a predetermined rate with the exception of sodium silicate and pine oil, which may be varied as froth conditions and control assays indicate. The pulp is not heated and flotation feed temperature during the winter averages about 48°F. Daylight has been eliminated from the flotation floor and fluorescent lighting installed throughout, providing ideal conditions for visual observation of the flotation froth. Quick assays for control are run hourly on tailing and every two hours on concentrate. Complete assays of feed, concentrate, and tailing for each shift are available within eight hours of the completed shift.

The flotation sump floor is at ground level. The cell floor is at an elevation 20 feet higher, allowing good launder slopes and a minimum of collecting sumps. All sumps are of concrete and all pumps throughout the operation are Canadian Allis-Chalmers *S.R.L.* with the exception of mill discharge and water sumps. The flotation floor is sloped to the centre, with a clean-up ditch running the length of the floor. Clean-up pumps and elevators provide a quick method of handling spills. A barrier divides the tailing and concentrate sections of the sump floor to prevent loss of concentrate in tailing or dilution of concentrate by tailing.

Screen analyses of flotation products, and power consumption, are given in Table IV.

TABLE III.—SCREEN ANALYSES OF GRINDING CIRCUIT PRODUCTS

Mesh	Rod Mill Discharge		Ball Mill Feed		Ball Mill Discharge		Classifier Overflow	
	Percent by Weight		Percent by Weight		Percent by Weight		Percent by Weight	
	This mesh	Cumulative	This mesh	Cumulative	This mesh	Cumulative	This mesh	Cumulative
				No.7 Shaft Ore				
+ 4	1.0	1.0	1.0	1.0	0.8	0.8		
+ 8	1.6	2.6	4.4	5.4	1.7	2.5		
+ 10	4.9	7.5	4.8	11.2	1.5	4.0		
+ 14	8.8	16.3	9.1	20.3	1.9	5.9		
+ 28	19.5	35.8	23.7	44.0	7.8	13.7		
+ 35	9.9	45.7	14.0	58.0	7.8	21.5	1.4	1.4
+ 48	6.6	52.3	11.0	69.0	8.1	29.6	3.9	5.3
+ 65	6.5	58.8	9.3	78.3	10.6	40.2	7.8	13.1
+100	5.9	64.7	4.5	82.8	10.3	50.5	7.6	20.7
+150	5.8	70.5	6.1	88.9	10.0	60.5	14.0	34.7
+200	4.1	74.6	2.0	90.9	6.5	67.0	8.2	42.9
−200	25.4	100.0	9.1	100.0	33.0	100.0	57.1	100.00
				No. 5 Shaft Ore				
+ 4	0.6	0.6	1.0	1.0				
+ 8	1.2	1.8	2.6	3.6	0.7	0.7		
+ 10	3.2	5.0	4.1	7.7	0.6	1.3		
+ 14	6.6	11.6	7.2	14.9	1.6	2.9		
+ 28	17.7	29.3	22.4	37.3	6.5	9.4	0.1	0.1
+ 35	9.5	38.8	15.7	53.0	6.7	16.1	0.9	1.0
+ 48	7.3	46.1	13.8	66.8	8.0	24.1	3.7	4.7
+ 65	7.8	53.9	10.2	77.0	10.6	34.7	7.8	12.5
+100	6.9	60.8	5.1	82.1	11.1	45.8	6.3	18.8
+150	7.1	67.9	6.7	88.8	11.7	57.5	15.3	34.1
+200	4.2	72.1	2.4	91.2	1.3	58.8	9.6	43.7
−200	27.9	100.0	8.8	100.0	41.2	100.0	56.3	100.0

Figure 4.—Flotation section,

Reagent Handling

Reagent storage rooms are located at the head end of the flotation cells, with the reagent head tanks and feeders on a sub-floor overlooking the flotation cells. There are four separate storage rooms, one for xanthate, one for lime, one for pine oil, and one for liquid sodium silicate and copper sulphate solution. The pine oil and xanthate rooms are ventilated by exhausting the air through ducts with a fan and explosion-proof motor. All reagents are added in the liquid state with the exception of hydrated lime, which is handled by a Fuller Kenyon conveying system to the head tank of a dry reagent feeder. Clean-up from the reagent feeder floor and storage rooms is collected in a central sump, from which it is pumped to the tailing sump.

Sand Fill Plant

The sand fill plant contains three 12-ft. quadruplex Dorr *F.X.* classifiers and is situated at the end of the flotation floor, adjoining No. 7 shaft headframe. The flotation tailing is laundered to a central sump and pumped to a three-way distributor ahead of the classifiers. Approximately 60 per cent of the

total tonnage of flotation tailing is recovered as a rake sand acceptable for mine fill. The rake sands are water-borne at 60 per cent solids through 6-in. rubber-lined pipe *via* No. 7 shaft to the No. 5 shaft workings, a pipe-line distance of 10,000 feet. The classifier overflow is returned to the final tailing sump for disposal. The sand plant may be partially or completely by-passed and will operate sixteen hours per day.

Tailing Disposal and Concentrate Pumping

Mill tailing is pumped approximately 4 miles to the 1,100-acre Copper Cliff mill tailing disposal area, situated between Copper Cliff and Creighton. The bulk copper-nickel flotation concentrate is pumped a further 3½ miles to the Copper Cliff mill for copper-nickel separation and subsequent smelter treatment. Due to the rugged terrain traversed, the concentrate pipe line reaches a maximum elevation 48 feet higher than the Creighton mill and then drops 98 feet during the final two miles to the discharge at Copper Cliff. Figure 5 is a schematic drawing showing the complete Creighton-Copper Cliff pumping system. Pumping stations No. 1, No. 2, and No. 3 handle Creighton concentrate and tailing pulps. From station No. 3, the Creighton tailing goes to the disposal area and the concentrate pulp is relayed through stations *E* and *C*, of the Copper Cliff tailing disposal system, to the Copper Cliff mill. Table V gives pipe line distances and changes in elevation between pumping stations.

Table IV

Mesh	Flotation Concentrate Percent by Weight		Flotation Tailing Percent by Weight	
	This mesh	Cumulative	This mesh	Cumulative
	No. 7 Shaft Ore			
+ 35			1.9	1.9
+ 48	0.5	0.5	5.8	7.7
+ 65	2.4	2.9	10.3	18.0
+100	7.8	10.7	12.9	30.9
+150	11.6	22.3	12.0	42.9
+200	9.5	31.8	10.1	53.0
−200	68.2	100.0	47.0	100.0
	No. 5 Shaft Ore			
+ 35			1.3	1.3
+ 48	0.1	0.1	4.5	5.8
+ 65	1.6	1.7	9.4	15.2
+100	5.8	7.5	12.6	27.8
+150	11.4	18.9	12.8	40.6
+200	9.6	28.5	10.6	51.2
−200	71.5	100.0	48.8	100.0

Power Consumption = 3.32 k.w.h./ton

Table V

	Pipe Line Distance	Change in Elevation
Creighton Mill to No. 1 Station	5,500 feet	+17 feet
No. 1 Station to No. 2 Station	6,965 "	− 2 "
No. 2 Station to No. 3 Station	7,250 "	+ 4 "
No. 3 Station to **E** Station	3,900 "	+44 "
E Station to **C** Station	9,950 "	−45 "
C Station to Copper Cliff Mill	6,450 "	−23 "

The trestle carrying the pipe lines is of wooden construction throughout and was built with a minimum 0.75 per cent slope to and from each pumping station in order that lines would be self-draining. The four miles of new trestle from the Creighton mill to the tailing disposal area carries two 8-in. concentrate pipe lines, two 13-in. tailing pipe lines, a 6-in. water line enclosed in an insulated wooden box, and a walkway. All lines are of wire-bound creosoted wood-stave pipes in maximum 16-ft. lengths; those for concentrate and tailing are tested for 110 p.s.i. pressure and the one for water is 75 p.s.i. One of each of the concen-

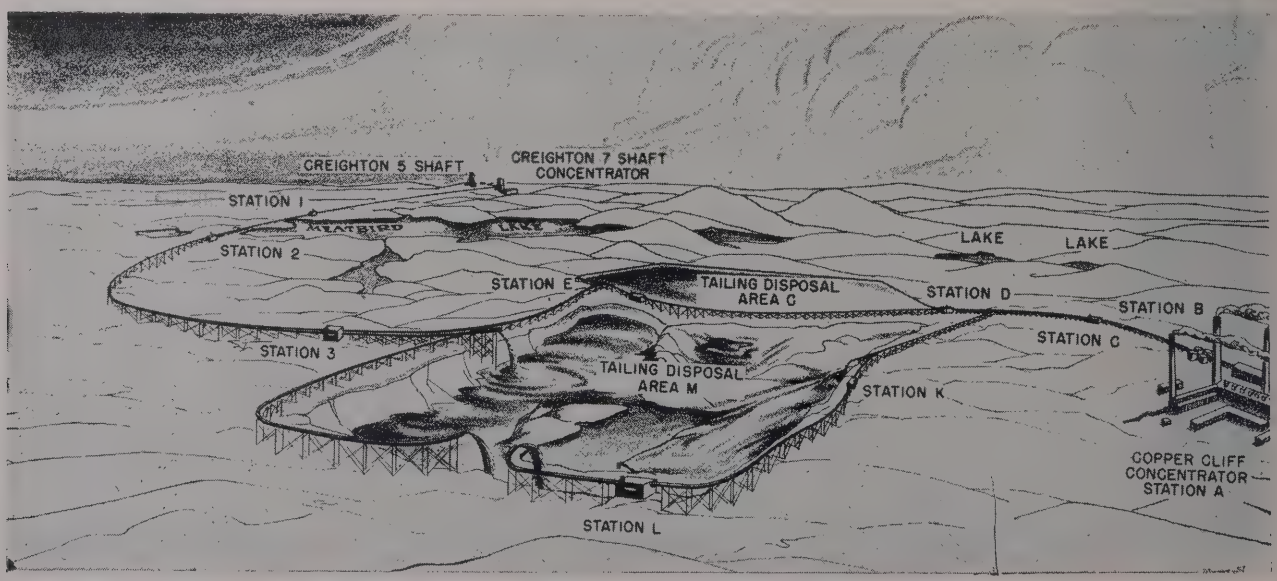


Figure 5.—Tailing disposal and concentrate pumping system.

trate and tailing lines is a spare line in case of emergency. From the point in the tailing disposal area where the Creighton tailing is dumped, the concentrate lines were installed on the existing Copper Cliff tailing trestle. The 6-in. water line supplies potable water to the stations and provides gland sealing and flushing water for the pumps and pulp lines.

Of concrete, steel, and cement-block construction, the pumping stations are completely fireproof. Sumps to receive tailing and concentrate pulps form one wall of the building. Three 8 in. by 6 in. pumps for concentrate, and three 10 in. by 8 in. pumps for tailing, all rubber-lined, are installed in a straight line. Auxiliary gland water pumps and a vertical clean-up pump in the concentrate drain sump complete the installation. The concentrate drain-back sump will hold three complete line drainages. Drainage from the tailing lines is through quick-dumping valves situated a short distance on each side of the stations.

The specific gravity of the concentrate and tailing pulps varies with the mill tonnage; pipe line velocity and volume handled must remain reasonably uniform. When milling 10,000 tons a day, approximately 1,800 tons of concentrate and 8,200 tons of tailing are produced. Under these conditions, the concentrate pulp has a specific gravity of 1.28 with a flow of 800 g.p.m. The 8 in. by 6 in. concentrate pumps are designed so one pump will handle this flow through one 8-in. line with a velocity of 5.11 feet per second. Original test work showed the loss in head due to friction when pumping 600 to 800 g.p.m. to be 17.5 feet of pulp per 1,000 feet of pipe. The longest concentrate pumping stage is a distance of 9,950 feet, where the static head is minus 45 feet. The tailing pulp has a specific gravity of 1.35 with a flow of 2,500 g.p.m. with the sand plant down. The 10 in. by 8 in. tailing pumps are designed so one pump will handle this flow through one 13 in. line with a velocity of 6 feet per second; the loss in head due to friction in the line approximates 11 feet of pulp per 1,000 feet of line.

The complete pumping system has been designed for trouble-free operation despite the rigorous northern winter weather. Several times the temperature has ranged to 30 degrees below zero, with no operating difficulties. The temperature loss of the concentrate pulp during extremely cold weather is approximately 2°F. per mile.

Sampling and Assays

Total metal input to the mill is determined through automatic sampling of the feed by a line of two Vezin samplers followed by a Roll crusher and a Snyder sampler. Samples of tailing are assayed every hour, samples of concentrate every two hours. Percent solids and sizing analyses of the various mill pulps are made regularly for operating control. The mill is equipped with a completely modern sample room and assay laboratory where assays are made on diamond drill cores and mine samples as well as on mill samples.

Mill Water Supply

Water is supplied from the Vermilion river through six miles of 20-in. continuous wood-stave pipe line. The line is covered with 2 ft. to 3 ft. of earth its entire length for protection from the wind. The pump installation at the river delivers 3,500 g.p.m. to booster pumps at the mill, where the pressure is raised to 60 p.s.i. for process water. The pumps at the river are operated by remote control from Creighton. A chlorinating unit on the line makes the entire water supply potable. Continuous recording flow meters record the total mill water consumption, the flow to each grinding unit, and the flow through the 6-in. line servicing the concentrate and tailing pumping system. Only fresh water is used at the Creighton mill; the water recovered from the concentrate and tailing is used to augment the Copper Cliff mill water supply.

Power

Power is supplied by the Ontario Hydro Electric Power Commission through the Company's 60-cycle distribution systems. It is transmitted at 44,000 volts and stepped down by a bank of transformers to 2,200, 550, 220, and 110 volts. All motor control equipment is in six switch-rooms conveniently located throughout the mill.

Change-House

The mill change-house, located on the second floor and reached by a ramp, has 130 two-compartment steel lockers. There are fifteen individual showers in the tile-lined group shower room. The lockers, lavatories, and shower room are serviced by a ventilation system which exhausts the air through the roof.

Miscellaneous

Supervision is accommodated in a well-lighted office with separate outside entrance, and with separate washroom facilities. This office is located on the ground floor adjacent to the first-aid room, the sample room, and laboratory. On the floor immediately above, overlooking the time-clock aisle, is the mill shift bosses' office, which has an individual locker room and washroom attached.

All major shopwork is done in the shops at Creighton No. 5 shaft or in the shops at Copper Cliff. For immediate operating repairs, a tool crib and mechanics' warehouse is located opposite the mill repair bay. A separate mechanical foreman's office is attached. Also located in this area is the electrical foreman's office with ample space for storage of wire, cable, and minor electrical repair supplies. General warehousing, with complete facilities for inside and outside storage, is at Creighton No. 3 shaft. This warehouse takes care of all supplies with the exception of heavy crusher parts, grinding steel, and flotation reagents, which are delivered direct to the mill building.

Steam for heating the building is supplied by oil-fired boilers separately housed a short distance from the plant. Unit heaters located throughout the building maintain a comfortable working temperature of +60°F. No steam is required in the milling process.

Further Potash Discoveries in Saskatchewan*

By A. J. WILLIAMS †

(*Annual Western Meeting, Calgary, Alta., October,* 1951)

(*Transactions, Volume LV,* 1952, pp. 160-161)

Introduction

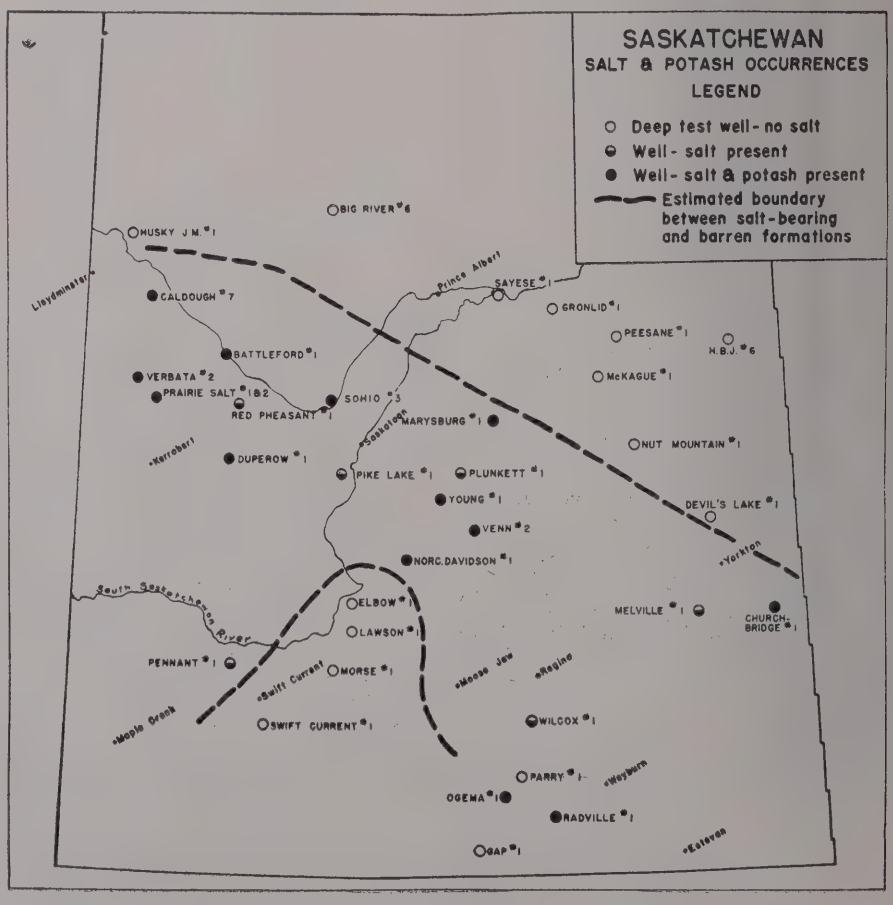
AN EXCELLENT paper entitled *Potash Discoveries in Western Canada* (1) was presented to the 1948 Annual Meeting of the C.I.M. by L. Heber Cole, of the Department of Mines and Resources, Ottawa. In the intervening three and a half years, the search for oil and gas in Saskatchewan has added considerably to the information available on the occurrence of potash. It is the purpose of this paper to summarize the additional data. For greater clarity, it may be useful, first, to recapitulate some of the highlights of Mr. Cole's paper.

Underlying the western plains of Canada there is a large deposit of sodium chloride, believed to be of early Devonian age. Associated with the sodium chloride are certain potash salts, primarily 'sylvinite,' a mechanical mixture of sodium chloride (halite) and potassium chloride (sylvite) crystals. The potash salts, when present, are usually near the top of the salt formation, though in at least one well they were found near the central part of the deposit. Cole reported on some seven wells in Saskatchewan which had encountered appreciable amounts of potash; these were the Ogema, Radville, Davidson Nos. 1 and 2, Prairie Salt Nos. 1 and 2, and Verbata wells. Depth of potash from surface ranged from 7,653 feet in the Radville well to a minimum of 3,466 feet in the Verbata well. Such depths do not compare favourably with those at which the potash is found in mines in the United States, France, and Germany, where about 1,000 feet is the average. The greatest concentration of sylvite encountered was in the Verbata well, where a bed 11 feet thick carried the equivalent of 21.64 per cent K_2O. This concentration compares favourably with that in potash mines being operated in other parts of the world. Since the salt bed apparently continue as far north as Waterways, Alta., where it was encountered only some 700 feet below the surface, it was hoped that potash also would occur farther north and at shallower levels.

Discoveries Since 1948

Drilling since 1948 has nearly eliminated that possibility, at least insofar as Saskatchewan is concerned. As indicated on the accompanying map, a string of deep test wells put down at intervals along and north of a line from Husky Jet Monnery No. 1 well north of Lloydminster to Devil's Lake No. 1 well near Canora have been found barren of salt although all pierced the zone in which salt and potash might be expected. A very tentative suggestion of the approximate northern boundary of this evaporite section as defined by our Department geologists is shown on the map. There is a similar barren section in the south and southwest part of the Province, as indicated by the six holes there which encountered no salt. It would appear, however, that the area between these two boundary lines, from Estevan to Lloydminster, is underlain by at least one major and continuous salt bed.

The discoveries since early 1948 of potash in that salt bed are shown by name and symbol on the map. Such pertinent information on these wells as has been released for publication by the oil exploration firms who supplied the core samples is tabulated in Table I. For the purposes of this paper, it is unfortunate that it is not possible to give the exact



*Presented with the permission of the Minister, Saskatchewan Department of Natural Resources, Regina.

†Director of Resources Utilization.

(1) *Transactions*, Vol. LI, 1948, pp. 83-92; BULLETIN, March, 1948.

depths of core taken in the most recent holes, but at least the approximate depths have been released from confidential status by the oil exploration firms.

Explanatory Notes to Accompany Table I

(1) Where analyses of more than one section in any particular well core are shown, the core samples have been taken consecutively, with no breaks except those for lost core, working downward from the uppermost potash showing.

(2) Since the taking of cores in the salt-potash zone has been incidental to oil exploration work, complete and satisfactory cores are not available in all wells shown in the table. The following brief comments will add to the value and meaning of the tabulated data:

(*a*) C. S. Battleford — Coring reasonably complete.

(*b*) Caldough No. 7 — Core reasonably complete.

(*c*) Red Pheasant — Only one foot of salt core taken, at extreme upper limit of salt bed.

(*d*) Sohio No. 3 — No core taken; sample of drilling mud indicated presence of potash.

(*e*) Duperow-Crown — Core recovery good in upper 16 feet of salt-potash zone.

(*f*) Churchbridge No. 1 — Upper eleven feet of salt bed cored with good recovery.

(*g*) Melville No. 1 — Major salt bed not cored; sample taken and analysed was from one of the minor salt seams overlying the main salt-potash deposit.

(*h*) Plunkett No. 1 — Major salt bed was not cored.

(*i*) Young No. 1 — Top of salt bed was not cored; there is some indication that up to 90 feet of salt and potash overlie the 10-foot section cored and reported.

(*j*) Venn No. 2 — Coring commenced after salt had been drilled for several feet; seven feet of lost core in 17-foot section cored and sampled.

(*k*) Marysburg No. 1 — Top 12 feet of salt section cored and sampled.

Table I.— Data on Recent Deep Wells in Saskatchewan Which Encountered Potash

Name of Well	Location					Depth of Top of Potash Zone from Surface, ft.	Thickness of Potash Zone	K_2O Equivalent %
	L.s.d.	Sec.	Tp.	R.	W. of M.			
C.S. Battleford No. 1	7	30	43	16	3	2,814 2,818	4′ 0″ 8′ 0″	4.49 1.61
Caldough No. 7	4	18	48	22	3	2,794	38′ 0″	Traces up to 1%
E. H. S. Red Pheasant No. 1	5	32	39	15	3	About 3,800	1′ 0″	Trace
Sohio No. 3	13	12	40	8	3	About 2,800	Drilled but not cored	Drilling mud contained 2.6 gm./litre KCl
Tidewater Duperow Crown No. 1	4	9	35	16	3	About 4,300	1′ 8″ 0′ 9″ 6′ 7″	19.0 7.8 1.4
Sohio Churchbridge No. 1	6	8	22	32	1	Between 2,800 and 3,100	1′ 0″ 1′ 0″ 0′ 9″ 1′ 3″ 1′ 0″ 1′ 0″ 1′ 0″ 1′ 0″ 1′ 0″ 1′ 0″ 1′ 0″	7.7 11.1 11.6 13.4 25.9 12.7 12.4 4.3 10.1 17.2 3.2
Sohio Melville No. 1	11	14	22	6	2	Between 3,400 and 3,600	5′ 0″	0.5
Sohio Plunkett No. 1	1	15	34	25	2	Between 3,800 and 4,000	2′	0.03
Sohio Young No. 1	15	11	32	27	2	Between 3,500 and 3,800	1′ 6″ 3′ 0″ 0′ 5″ 5′ 0″	7.2 3.6 6.2 3.4
Sohio Venn No. 2	4	8	29	24	2	Between 3,400 and 3,700	5′ 0″ 0′ 6″ 6′ 6″	1.2 19.9 1.2
Sohio Marysburg No. 1	16	29	38	22	2	Between 2,700 and 3,000	1′ 5″ 0′ 9″ 1′ 3″ 4′ 3″	1.7 5.5 10.0 1.63

Summary and Conclusion

In summation, it can be seen that the Verbata No. 2 well still shows the richest, most mineable occurrence of potash in the Dominion. On the other hand, recent discoveries have indicated considerable continuity of the potash layers associated with the main bed of underlying salt, and at least two new areas may bear further investigation. At Churchbridge, the potash zone is several hundred feet closer to the surface than at Vera, and further drilling might reveal a rich deposit in that area. At Young, the fact that small potash values were found well below the upper portion of the salt bed could point to a thicker-than-usual potash formation of possible commercial interest.

We can conclude, in much the same vein as did Heber Cole, by saying that further drilling must be done in specially selected localities to outline the potash zones and to determine if the potash exists in deposits of commercial size and quality. One company is presently operating on an exploration permit covering about 100,000 acres in the Vera area, and it is expected that it will carry out such further drilling in that area within the near future. In any case, present information indicates the existence in the Province of vast quantities of potash, which at the very least are a valuable addition to the known world reserves of this important mineral.

A Canadian Integrated Coal and Steel Industry

By L. A. FORSYTH[*]

(*Ottawa General Meeting, January,* 1952)

(*Transactions, Volume LV,* 1952, pp. 162-163)

SINCE I received and accepted your invitation to present to this meeting a paper on *A Canadian Integrated Coal and Steel Industry,* I have given the subject a great deal of intensive study.

Two conclusions have emerged from my considerations: first, that the subject should be dealt with by a person having a firm grasp of its technical aspects; and second, that it is a subject which requires expert analysis of the complicated problems it presents in pure economics.

A third conclusion inevitably results from the two just mentioned — namely, that the mantle of the prophet which has been draped upon my shoulders is too large for me to wear, because I am neither a practical nor a theoretical coal and steel man; and I am not an economist.

With the warning that I have discovered, and am keenly conscious of, the inadequacy of my background for the task, I shall attempt to place before you those aspects of my subject which appear to highlight it from a layman's point of view.

In the first place, in order to avoid confusion as between my audience and myself, some definition of terms may be required.

As I interpret it I am discussing — a coal and steel industry, using the last word in the generic and not the unit sense, self-sufficient in the provision of all its needs from within the geographical limits of Canada.

Now, I start with the proposition that, in the very broadest aspects of such a concept, the Corporation of which I happen to be the executive head meets my definition as a combination of operations resulting in one fully integrated unit of a coal and steel industry.

DOSCO has, in the Province of Newfoundland, deposits of limestone and iron ore which it quarries and mines and transports to the Canadian mainland; and at Sydney, in the Province of Nova Scotia, this ore and this limestone are processed with coke, produced from coal mines in close proximity to our steel plant, through the operation of blast furnaces and basic open hearths to obtain steel in its primary ingot form.

I shall make no attempt to describe in any detailed fashion the subsequent operations by which, at Sydney and at our other plants in Nova Scotia, New Brunswick, Quebec, and Ontario, the primary steel is converted to semi-finished, finished, and fabricated steel products of a rather remarkable and certainly numerous variety.

I should say at once that, notwithstanding the variety of end products produced by *DOSCO,* we do not begin, nor do we attempt, to fill every demand for finished steel products.

Candour compels me to state also that in order to achieve the desired efficiency in our open hearth operations we have, to date, found it desirable to import from abroad a small supply of premium quality lump ore. This is a circumstance which may well cease to concern us as and when the production of the newly discovered Canadian deposits becomes available.

The independence of *DOSCO,* so far as its supply of raw materials is concerned, from importations derives of course in the first instance from the fact of the strategic location of ore, limestone and coal within the radius of economic transportation to the point of manufacture.

Subsidiarily, our operation as an integrated unit in the production of primary steel stems from our ability to produce the raw materials I have mentioned at a cost which permits of their ultimate disposal in finished form in the available markets.

I mention these features of our business because in one aspect or another they have a very important bearing on the discussion of an integrated industry from a national point of view.

Before I leave the *DOSCO* picture for the wider scene, I should, I think, direct your attention to the fact that the very vastness of our country and the concentration of steel consumption which characterizes it today emphasize transportation and the cost of it as our major economic problem.

While, in the United States, there undoubtedly are large units of the steel industry, a portion of whose output goes into consumption at equally long distances from its point of origin, there is no one which must find an ultimate market for virtually its entire production at points so remote from the source as *DOSCO* is compelled to do.

Coming now to consideration of the Canadian steel industry as a whole, what is the raw material situation?

I dismiss perhaps too readily the question of limestone as not being one of major concern because my information is that abundant deposits of this essential material occur at intervals not too widely spaced throughout the expanse of our country from east to west.

Iron ore of satisfactory quality and in quantity sufficient to supply the Canadian industry in any degree of expansion presently predictable exists in proven deposits in the Provinces of Newfoundland, Quebec, and Ontario. In British Columbia, smaller deposits are known to exist, and in the treatment of other ores there are sources of iron which can be made available as and when required.

With respect to coal, the situation is less reassuring. The principal coal deposits in Canada are found, broadly speaking, in the extreme east and west — in the Provinces of Nova Scotia, New Brunswick, Saskatchewan, Alberta, and British Columbia.

While, in quantity, the known coal reserves of the country are colossal, from the standpoint of the steel industry and its present practices, the proportion of metallurgical coal, so-called, is relatively small and not too well-located.

[*] President, Dominion Steel and Coal Corporation, Limited.

Three-quarters of our installed capacity for steel production is located in Central Canada and our only known supplies of coal suitable for use in the manufacture of steel are found virtually on the Atlantic and Pacific seaboards. Certainly, as things stand at present, coal of the type required is obtainable from the United States under conditions with which Canadian coal could not possibly compete.

Here, again, is the transportation factor. I cannot myself visualize any solution of that problem which will enable the steel industry of Central Canada and the coal production either of the east or of the west to become integrated upon any basis of sound economic principle. Failing, therefore, the discovery of coal of satisfactory quality in either Quebec or Ontario, the establishment of a fully integrated steel and coal industry in Canada appears to be unlikely of achievement.

I do not overlook the fact that intensive research, in which some progress has been made, has been directed toward the problem of eliminating coal as an essential ingredient in the manufacture of steel. I cannot, therefore, dismiss as irrelevant the fact that, as a result of the construction of the projected St. Lawrence Seaway, hydro-electric power in large quantities will become available; and that such power may contribute in no small measure to the manufacture of steel in this country.

However, this potential does not operate toward the integration of coal and steel.

One further observation upon the raw material situation appears to me to be pertinent. According to present practices, and to meet the requirements of the steel industry today, substantial quantities of steel scrap must be available. I have not made a close study of the statistics, but I have grave doubts as to whether Canadian plants could have attained, and whether they can maintain, present production not far below 3,500,000 tons, to say nothing of a possible 4,250,000 tons projected for 1952, in absolute independence of scrap supplies from abroad.

Now there is a growing spirit of nationalism, or perhaps in certain manifestations it is better described as provincialism, using the word in the sense of geographical limitation.

In many quarters there is an urgent insistence upon the processing of all raw materials of Canadian origin within Canada. Sometimes it goes to the extent of emphasis upon the desirability of processing the primary products of one Province within that Province.

To the proposition as a theory I have no objection; but I do see grave obstacles to its practice. Slavish and uncompromising adherence to the narrower or provincial phase of the doctrine could obviously present a series of impasses within our country which could perhaps only be overcome by some such expedient as the enlargement of the boundaries of one province to include the whole or at least the strategic portion of another.

In the wider national field, the implications of such a doctrine carried to its ultimate conclusion raise questions of international political and economic problems, which I do not feel competent to forecast or to analyze.

I do have a shrewd suspicion, to which keener students of international affairs than I have given some support, that national boundaries have been changed following upon armed conflict having as its originating causes the location of important natural resources in one country and the dire need of them for the economic health of another.

I know that the demand for iron ore from Newfoundland in countries other than Canada is one factor which enables *DOSCO* to set up an operation which lowers its cost of ore for its own operations. In effect, the benefit which our surplus supply confers upon our export market is reflected in an economic advantage for a Canadian unit in the industry.

The attention of many Canadians has been focused upon the iron ore discoveries of the Labrador and the potentialities inherent in their development from a Canadian point of view.

Whether it would be possible or even desirable to take steps to prevent the export of the production from these deposits and insist upon processing it in Canada is not only a matter of internal politics but of great importance in the field of international relations.

My own view is that, as now appears to be the case, the demand for steel in Canada has not yet reached the point where any apprehension should exist as to the adequacy of our ore supplies. On the other hand, at the present level of demand for steel, the supply of Canadian coal of the required quality is inadequate for our steel industry as presently located.

There is every indication that, with the growth of Canada, the demand for steel will increase, and with that increase there will develop a necessity for further reliance upon a supply of metallurgical coal from non-Canadian sources. And it may well be doubted whether, even if transportation problems could be overcome, our supplies of metallurgical coal would be sufficient in quantity to smelt the available ore.

The economically efficient development of our iron ore deposits imperatively demands a planned output in excess of our capacity to use it. Full and complete integration of our coal and steel facilities appears to be incapable of immediate realization, if ever.

It would seem that a policy which forestalls the possibility of failure of ore supplies for the Canadian steel industry — due regard had to the tempo of expansion — while offering a surplus for export against our requirements of coal and other commodities which we must import, is the only one which will enable us to make immediate use for our national steel industry of our newfound wealth in iron ore.

I fear that the foregoing is a somewhat diffuse and disconnected discussion of the subject you have assigned to me. If it has any merit, it is that it is an attempt to be realistic and objective in respect of problems which are too often approached with more emphasis upon idealized than upon practical concepts.

What I have endeavoured to do is to make the point that, where the raw materials are so located as to create advantage in production of the resultant product, namely, at our eastern and western extremities, this advantage is lost by reason of the cost of transportation of the product to the market; and that, when the industry is advantageously located marketwise, the use of materials wholly Canadian in origin is defeated by the cost of carriage of the material, in the main coal, to the point of production.

Western Coking Coal Resources

By T. G. EWART

(*Ottawa General Meeting, January,* 1952)

(*Transactions, Volume LV,* 1952, pp. 164-166)

THE COAL RESOURCES of this continent represent over 90 per cent of the total fuel reserves, but the annual consumption of all fuels is so distributed that only 50 per cent is drawn from coal and the remainder from natural gas and oil. However, the requirements of our modern life are making substantial demands on certain types of coal, such as the coking and smokeless kinds, and these grades are being depleted at a relatively fast rate.

It is comforting to know that there are almost unlimited quantities of lower rank coals in this country and on this continent, for the day when the relatively meagre reserves of natural petroleum and natural gas are finally depleted. Their replacement in the fuel market by liquid fuels made from coal will ensure that our oil and gasoline 'civilization' can be expected to continue without a revolutionary change, but the increase in coal consumption on our present schedule of energy requirements would be of the order of three times that presently produced.

With these opening remarks on the potential importance of coal for the fuel requirements of the country, let us examine its more particular position in regard to the metallurgical industries. The great steel industry of this continent owes its existence in no small measure to the large deposits of valuable metallurgical coal in the Appalachian area, which have supplied the bulk of American steel industry requirements for the past century. Even the Canadian primary steel industry, due to its geographical location, has had to rely on obtaining about three-quarters of its coke requirements from that source or from coal imported from that area.

In the past few years considerable anxiety has been expressed by mining engineers and fuel technologists regarding the comparative exhaustion of this once prolific source of good-grade coking coal and also of low-volatile coals used in making suitable blends with high-volatile coals. According to Fieldner:

> "The actual available reserve under present mining and metallurgical practice is not known, but it is known that many of the best beds in Pennsylvania have been exhausted and that the steel industry is beginning to use notable amounts of lower-grade coking coals which require the best possible cleaning to reduce the sulphur content to a reasonable amount.
>
> "The future calls for much research to develop more effective methods for removing ash-forming material and sulphur from low-grade coals not amenable to present methods of preparation. This will be required for utilizing the high-sulphur coals of southwestern Pennsylvania, northern West Virginia, and Ohio in the Appalachian region, and to a much greater extent in the Interior Provinces.
>
> "Likewise, the depletion of the beds of strongly coking coal makes it necessary to increase the practice of blending high-rank with low-rank coking coals before the limited reserves of high-rank coals become exhausted. Fundamental research on the chemical and physical mechanism of the coking process also will aid greatly in finding means of making good coke from the larger reserves of poorly coking high-oxygen coals of the Interior Provinces.
>
> "The most disturbing factor in the coking coal-reserve situation is the limited supply of low-volatile bituminous coal essential for blending with most high-volatile coals for making a good metallurgical coke. Only 4 per cent of the bituminous-coal reserves of the United States consists of low-volatile coal; increasing proportions (three times the amount used for coke production) of it are being used for fuel purposes because of its semi-smokeless properties. Its comparatively early exhaustion will require the development of a substitute, which probably can be done partly, at least, by the low-temperature carbonization of high-volatile bituminous coal".

The utilization of coking and low-volatile coals for uses other than coking is, of course, contributing to the depletion of these premium fuels. Coking coals are used extensively by the railways and low-volatile coals find much application for heating in cities with strict smoke ordinances. Recent U. S. Bureau of Mines estimates indicate that under one-half of the coking coal mined is used for the manufacture of metallurgical coal and about a quarter of the low-volatile coal is used in coking. The Canadian situation is very similar, with a large proportion of the coking coal mined being used by the railways and by industry.

A seam-to-seam survey is presently being carried out by the U. S. Bureau of Mines in the Appalachian area with a view to establishing the precise mineable reserves of seams down to 14 inches in thickness. This survey will be extended to the seams of coking coal in the Mid-Continent and the West.

From a coking coal viewpoint, the coal resources of the United States are distributed rather unevenly. The eastern states, with 20 per cent of the known coal reserves, possess the bulk of the coking coals. In the mid-continent, which accounts for another 20 per cent of the coal reserves, there are some good coking coals but the majority are weakly caking, with high sulphur content. The northern great plains and the western states have 60 per cent of the total reserves but three-quarters of the coal is lignitic or sub-bituminous.

A survey conducted by the United States Geological Survey of the coking coals in the west has indicated that there are some twenty-

*The Crow's Nest Pass Coal Company, Limited, Fernie, B.C.

one localities where coal of coking properties is found, but only four areas have sizeable reserves. No precise figures are available of these reserves but a very rough approximation from available data would indicate that there are mineable reserves of reasonable quality coking coals of the order of two to three billion tons. At the present time, the Sunnyside Castle Gate field in Utah, with reserves of the order of 150 million tons, is the principal supplier in the United States west, with an annual production of about 2,000,000 tons of coal. The coke from this field is in great demand for the growing steel industry in Utah and California. Compared with this deposit there are fairly large deposits in southern Colorado and northern New Mexico, possibly amounting to 1½ billion tons of coal with good coking properties. Closer to the Canadian border there are several deposits of coking coal in Washington State and in Montana, but these would amount to no more than possibly 500 million tons in reserves.

On the Canadian side there is a considerably greater abundance, and more concentrated areas, of coking coals. From the International boundary for 350 miles north, an almost continuous belt of coal extends along the continental divide between Alberta and British Columbia. A very large proportion of the coal is of coking quality though there are also areas of weakly non-caking low-volatile coals. This belt contains nearly nine-tenths of the bituminous coals of Canada and about half of it is of coking quality, mainly of medium-volatile rank. It is interesting to note that by far the greater proportion of coke in the United States is made from coal to high-volatile rank, whereas in Western Canada most of the coking coal is of medium-volatile rank. The common practice of blending low-volatile coal with high-volatile coal is tantamount to preparing a medium-volatile coal. Many medium-volatile coals of suitable grade produce coke of equal quality to that of prepared blends of similar volatile matter content.

The recoverable reserves of coking-quality coal in Western Canada including the Yukon and Northwest Territories are evaluated at approximately 18 billion tons, but not all of these can be regarded as coking coals of recognized grades for metallurgical use. Many are high in ash and cannot be compared in coking properties with the well known coking coals of the Appalachian region. However, there are some coals of moderately low ash and good coking properties in the Crow's Nest area and the largest known deposit, containing about 4 billion tons of recoverable reserves, is located just west of the Alberta-British Columbia boundary, known as the Fernie basin. This compact basin is very prolific in coal, with twenty-four seams aggregating about 170 feet in 3,000 feet of strata. A large proportion of the coal is of good coking quality and the estimates of reserves only account for the fringes of the basin to a depth of 2,500 feet. There are considerable quantities of coal in excess of this figure but at greater depth.

There are other areas of coking coal, as for example on Vancouver island, but by comparison they have not the significance of the known deposits along the Rocky Mountains belt. There are also indications that there are substantial deposits in the less explored northern areas, but the Crow's Nest area, particularly the Fernie basin, will be a continuous source of good grade coking coal for many years to come. Its favourable geographical location, with good communications along the main line railway from west to east and a nearby link to the south, places it in a favourable position to supply not only a western and central Canadian market but also a demand south of the border, where the significance of these coking coal reserves apparently is not fully realized. The present production of coke, of the order of 300,000 tons per annum, can be, of course, considerably expanded if the market conditions are favourable. These deposits occur mostly in a tectonically disturbed region which is conducive to a high-ash incidence in the coal. However, some of the Fernie Basin coals are remarkably low in ash, of the order of 5 to 6 per cent. In any event, all the principal sizes of the coals of this area are cleaned in preparation plants.

Generally speaking, the selection of coal for coking is determined to some extent by economic considerations, and the demand for coke with certain physical properties is balanced against the availability of various coals that, separately or in blends, can be prepared to yield coke possessing these properties. No complete set of standards for the identification of good coking coals by chemical and physical properties exist. The rank of the coal is not the only criterion, as many coals of apparently suitable rank have poor coking qualities. It is believed that the original position of the seam is an important factor in determining coking properties. The coals of the Crow's Nest area are strongly coking and can produce a dense coke if carbonized in a slot-type by-product oven. However, no such installation exists in the West, the coke being made either in Beehive ovens or in the horizontal-type sole-heated ovens. The coke produced from the latter type of oven under these conditions is not entirely satisfactory for high-shaft iron blast furnaces, due to its low density. In any case, there are no such blast furnaces in the West and the largest demand in the immediate markets around the Crow's Nest area is for coke for non-ferrous metallurgy and for foundries. Aside of the ash content, which can be controlled to some extent by the degree of cleaning carried out, the coals are quite low in sulphur (mostly under ½ per cent) and in phosphorus. The ash fusing point is moderate to high, being over 2,500°F. With steady market demands, there would be an incentive for the coal companies to produce a constant grade of coke which the user could depend on in adapting it to the requirements of his process.

In view of the present size of the carbonizing industry in the Canadian west, it is too early to discuss the resultant benefits of a sizeable by-product industry in relation to the production of the important by-products such as benzols and tars, but unquestionably these products would be absorbed in local markets, either in their crude state or, as industrialization expands, as source materials for a chemical industry. Demands for really substantial quantities of coke would arise only if there were a large-scale steel industry in the West. While there are no known large-tonnage deposits of iron ore in the West, it would not be unreasonable to assume that ore would move to meet the fuels and flux materials, available in such abundance in this area, when increased demands for steel products, developing concurrently with the expanding industrialization and increase of population of the west, warranted it. These may be the circumstances that will prevail in several decades from the present time. Meanwhile, the Crow's Nest area of Alberta and British Columbia can furnish the progressively growing demands for coke both for Western Canada and in the coke-deficient areas of the western United States.

Discussion

In reply to a question concerning the sulphur and ash content of coking grades of coal available in Alberta, it was stated that the range for sulphur is from 0.45 to 0.6 per cent, for ash 7 to 7½ per cent, and for volatiles 26 to 27 per cent.

Dr. T. B. Williams, Controller, Coal, Petroleum and Natural Gas, Department of Lands and Forests, Province of British Columbia, commented that, whereas it was mentioned in the paper that coking coals have been found in an area extending from the International boundary for 350 miles in a north-westerly direction, the fact is that discoveries extend over twice that distance, into the northeastern part of the Province of British Columbia.

He mentioned that the Mines Branch in Ottawa, the Pacific Great Eastern Railway, the Canadian National Railways, the Canadian Pacific Railway, and certain power plants have tested coal from the Pine River area. Because of its high B.t.u. value, up to 15,000, and its low ash content, it should be suitable for use in the proposed coal-fueled fire-jet locomotive engines now being tested.

He said that indicated coal reserves in the tested area of the Pine River valley run as high as fifty million tons in seams of four or more feet in thickness. Not all of this is coking coal. A typical analysis of the best available coal from this district shows that it carries from 2 to 6 per cent moisture, 3 to 5 per cent ash, 19.8 per cent volatile matter, and 74.1 per cent fixed carbon, and that it has a B.t.u. value of approximately 14,710 and an ash-fusion point of 2,780°F. The Federal Mines Branch reported the coke as 'good'.

The coal field may be traced into the area north of Peace river and for a long distance to the southeast. There is reason for believing that the actual tonnage far exceeds that mentioned above.

Iron Ore Supply: Present and Future

By W. M. GOODWIN *

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Introduction

IN THE preceding paper we have learned about Canada's resources of metallurgical coal. There is an adequate amount in Nova Scotia, a very large reserve on the eastern flank of the Rocky mountains, and a strictly limited supply on the Pacific coast. Added to these is an unknown quantity of coking coal in the Peace River district, at present rather isolated but possibly capable of industrial development in the future.

The chief use of metallurgical coal is to make iron and steel. For this, an adequate supply of iron ore is required. So we should now scan the Canadian supply of iron ore, the possibility of its increasing use at home, and the probable markets abroad.

Trade in iron ore tends to be international, as is the case with many other raw materials. The present tendency with iron and steel, in contrast, is to keep its manufacture within national boundaries, and there is no evidence that this will change much in the years to come. We should keep these two points clearly in mind when considering the disposal of iron ore from Canadian mines. We should remember, also, that the modern blast furnace and steel plant has such a large output that a correspondingly large population, reasonably compact, is needed to consume its products.

Large Ore Supply in East, Small in West

Canada has now four major iron ore fields, well situated to serve adequately the eastern half of the country. Wabana and Labrador are well placed to serve overseas markets and plants on the eastern seaboard and eastern interior of the United States and Canada. Algoma and Steep Rock are an integral part of the great Lake Superior iron ore pool. Each of these will be discussed below in some detail. The magnetite deposits of eastern Ontario and the adjoining part of Quebec now give promise of becoming a fifth major field. The ilmenite deposits of Allard lake have commenced a substantial output of iron in conjunction with titanium concentrate. The sulphide ores of Sudbury and Noranda will give a still larger output of iron when suitable processes have been worked ont.

The western half of the country is not so well served with iron ore, so far as is known at present. Production for export has been commenced from the comparatively small magnetite deposits of the Pacific coast islands, and this will help to provide an economical supply of ore for the smelter at Vancouver, recently announced. No sedimentary deposits of iron ore have been found as yet in the prairie region, which contains such a large part of Canada's other principal natural resources. Manitoba might well be served, however, by a blast furnace drawing its coal from Alberta and its ore from Steep Rock, when the present rolling mills reach the limit of available scrap. No evidence of iron ore has yet been found in the vast hinterland of the prairies, which comprises the western half of the Precambrian Shield. The eastern part of British Columbia has hematite at Kitchener and a large potential supply of iron from the pyrrhotite tailing of the Sullivan mine, which can be used in due course to feed a smelter in Alberta.

There is an interesting situation in Yukon, where evidence is accumulating of an iron range, or possibly a series of ranges, extending for about 500 miles in a great arc from the Mackenzie mountains northwestward to a locality near the Alaska border north of Dawson. While no orebodies have been found up to the present, there is now an incentive to search for them, in spite of the long distance (about 400 miles) to the port of Skagway. It seems possible that this range might provide the first major reserve of iron ore on the Pacific side of the American continents.

The Iron Ore Trade

The broad features of supply and demand for iron ore in Canada are set forth in the accompanying tables. In Table I, the output of Wabana is included throughout, though Newfoundland did not join the Confederation until 1949. In 1940, Algoma's Helen mine began to produce substantially. During the next few years, the *U*-boat attacks curtailed seriously the Wabana shipments. In 1945, Steep Rock commenced shipments. In 1950, the Helen production was doubled to one million tons. In 1951, preparations were begun for a much increased production from Wabana, and for production from the Hogarth mine at Steep Rock Lake. Four deposits of magnetite in the Grenville rock series between Peterborough and Ottawa were proven by drilling and preparations have been commenced for production from one of them. The 360-mile railway into the heart of Labrador is under way. With these points in mind, estimates from 1952 to 1960 have been included, and some details are presented in Table II. The estimates for consumption allow for the Dofasco blast furnace recently completed in Hamilton, the Stelco furnace now under construction, the furnace for Algoma announced recently by Sir James Dunn, and possible additional furnaces in the years to come, as well as for the iron from the Sorel titanium smelter.

Reserves of ore are set forth in Table III. These figures will be noticed in detail when the separate properties are discussed. In each case, the reserve is ample to ensure a long life for the mine.

*Quebec Metallurgical Industries, Limited.

At first glance it seems anomalous that, during recent years, there should be so much iron ore imported when a somewhat similar amount is exported. The reasons are partly geographical and partly metallurgical. Wabana, with its abundant reserve and low costs, is in a position to export to overseas markets whenever these are available, as well as furnishing practically the whole of the Company's own supply at Sydney. The furnaces in Ontario, on the other hand, find it more economical to use only a comparatively small proportion of Steep Rock and Helen ores and to bring the larger part of their requirements from the United States, thus leaving the bulk of the high-grade ore from the Canadian mines available for export.

The international exchange of Lake Superior ore is likely to continue much as at present, with more and more of the Canadian ore available for American furnaces as the output of Steep Rock and Algoma increases. Overseas contracts for Wabana ore ensure a marked increase in production for several years, while the forthcoming shipments from the Labrador field will go almost entirely to the United States.

For many years, the Canadian iron and steel industry has subsisted on coal and iron ore from the United States. The balance of trade in iron ore will soon be reversed and we shall be in a position to repay gradually this debt to our friendly neighbour. The furnaces of Ontario will, however, always remain completely dependent upon American collieries.

An interesting point is the possibility that the low-silica, low-phosphorus ore from Labrador will enable the Sydney plant to make an advantageous mixture of ores to reduce the cost of its steel. The Wabana ore, alone, requires an abnormal volume of slag both in the blast furnace to remove silica and in the open hearth to remove phosphorus. A mixture of ores would reduce the volume of slag required, and the saving in fuel might offset the higher cost of the Labrador ore.

The Iron Ore Fields

Each of the principal ore fields and mines will now be reviewed briefly. They are rather a mixed lot, including several mines now in operation, some being prepared for operation, some awaiting development, and some still in process of exploration. A large part of the specific information is in the tables.

Table I.— Iron Ore — Canada (including Newfoundland)
(long tons)

Year	Total Production	Charged to Blast Furnaces			Domestic Prod. as a Percentage of Total Ore Charged	Domestic Ore Charged as Percentage of Total Ore Charged
		Domestic	Imported	Total		
1939	1,514,356	559,615	758,337	1,317,952	114.9	42.5
1940	1,779,115	818,493	1,273,219	2,091,712	85.1	39.1
1941	1,642,927	990,987	1,427,842	2,418,829	67.9	41.0
1942	1,352,779	861,229	2,364,389	3,225,618	41.9	26.7
1943	1,383,812	1,079,467	1,829,864	2,909,331	47.6	37.1
1944	1,078,587	794,936	2,323,983	3,118,919	34.6	25.5
1945	2,002,623	868,306	1,840,135	2,708,441	73.9	32.1
1946	2,618,682	795,744	1,459,678	2,255,422	116.1	35.3
1947	2,993,892	899,312	2,380,130	3,279,442	91.3	27.4
1948	2,911,731	931,792	2,559,826	3,491,618	83.4	26.7
1949	3,281,336	988,616	2,445,371	3,433,987	95.6	28.8
1950	3,218,983	1,248,850	2,477,501	3,726,351	86.4	33.5

Note: Total shipments from Wabana Mines, Newfoundland, are included in the "Total Production" column.
Shipments from Newfoundland to Canada are included in Domestic ore charged to blast furnaces.
Imported ore charged to blast furnaces excludes Newfoundland.

Table II.— Estimates of Future Production
(millions of long tons)

	1951	1952	1953	1954	1955	1956	1960
Wabana	1.7	2.5	2.5	2.5	2.5	2.5	3.5
Labrador	—	—	—	2.5	5.0	10.0	15.0
Algoma	1.2	1.2	1.2	1.2	1.2	1.2	.25
Steep Rock	1.3	1.2	1.2	2.0	3.0	3.0	4.0
Grenville Area	—	—	0.4	0.8	1.2	1.2	2.0
	4.2	4.9	5.3	9.0	12.9	17.9	27.0

Table III.— Estimates of Ore Reserves
(millions of long tons)

	Proven	Probable	Possible	Total	% Iron	Content of Iron
Wabana (recoverable)	50	100	1,000	1,150	51	612
Labrador	400	400	2,200	3,000	55	1,650
Algoma	250	125	100	475	33	157
Steep Rock	120	240	360	720	52	374
Allard Lake (ilmenite)	200	—	—	200	33	66
	1,020	865	3,660	5,545	51	2,859

Wabana

The present method of mining the sedimentary beds of Wabana is room-and-pillar, which leaves about half the ore as pillars. Until the method is changed, one can count only half the ore as recoverable reserve. The possible area of ore-bearing strata is 70 square miles. If the three beds now being mined extend over this area with the same average thickness and grade as at present known, the possible ore is 2,500 million tons. Assuming half of this recoverable and deducting the 50 million tons already extracted, the total possible reserve is 1,200 million tons.

The quality of the Wabana hematite is unusually uniform, so far as it has been opened up, and there is no indication that it will change. The silica occurs mainly as silicates of iron and the phosphorus as shell fragments converted to calcium phosphate. No way of reducing the proportions of silica and phosphorus economically has yet been found.

The four mines now in operation are being re-equipped to give a greatly increased capacity. Contracts with Great Britain and Germany account for the estimated increased production commencing in 1952. The new equipment will permit a still larger output. The requirement at Sydney might increase somewhat. There is little likelihood of Wabana ore becoming acceptable to furnaces in the United States. A substantial expansion of output depends mainly upon increased orders from Great Britain and Germany.

Labrador

This new district is capable of supplying an abundance of high-grade natural ore, comparable to the Mesabi range at its best. As little stripping is required and as much of the ore stands above the general level as ridges, the output can be increased very rapidly, once the railway and port are in operation. Thus the Labrador field will be able to meet promptly any sudden demand from Canada or the United States, and from Great Britain and Germany, which can be served just as readily as the eastern half of this continent. Access to the Pittsburgh-Youngstown area will be hampered until the St. Lawrence Deep Waterway is provided. Meantime, a limited tonnage can be handled by railways from the ports of Montreal and Baltimore and by small lake freighters.

Already, the Iron Ore Company of Canada is investigating the economy of treating some of the ore of lower grade, which is available in abundance in Labrador as on the Mesabi and other Lake Superior iron ranges. The practice of washing low-grade ores was established over forty years ago on the Mesabi, long before there was any obvious need for it, and has been a boon to the whole district. It might prove to be an equally foresighted policy in Labrador.

The Company's large tonnage of manganiferous iron ore might become important in case of an emergency in the supply of this strategic metal, particularly if attempts now under way to make high-grade ferromanganese from such ore are successful.

Sorel

The initial difficulties with the large electrical reduction furnace of the Quebec Iron and Titanium Corporation at Sorel have now been overcome, and it is expected that the full complement of five furnaces will be installed in rapid succession, to give an output of 500 tons of iron a day from treatment of 1,500 tons of ilmenite. No further power is available to the Company from the St. Maurice river; but, if the demand for titanium dioxide and titanium metal continues to increase as at present, it seems likely that the Sorel plant can be expanded with the use of power from Beauharnois and later from Lachine, on the St. Lawrence. The deposits at Allard lake are ample to support a greatly enlarged output of ilmenite.

Algoma

The Helen mine of the Algoma Steel Corporation is the only siderite deposit in the Michipicoten area at present equipped for production. Its ore reserve now stands at 90 million tons. The underground mine is now producing ore at low cost with the aid of block caving and conveyor-belt hoisting. The sinter plant at Wawa, with nominal capacity of one million tons, produced 1.2 million tons of sinter in 1951. Recently, Sir James Dunn announced the discovery of a faulted extension eastward of the orebody, which is called the Alexander deposit, similar in size and grade to the adjoining Victoria and Helen sections of the range.

Algoma Steel's Siderite Hill deposit, with 100 million tons in reserve, is ready for exploitation on a similar scale, and the Britannia deposit, with 20 million tons, could support an output of about ¼ million tons of sinter a year. The Ruth and Lucy deposits of Jones and Laughlin have 40 million tons reserve and could support a yearly production of about ½ million tons of sinter. The Algoma sinter contains about 3 per cent manganese and is self-fluxing, which aids in marketing it. Though the sinter is much in demand, the capital expense of the necessary plant is heavy, and this has retarded development of the district.

Eastern Ontario

The marked success of the magnetic iron-ore mines in the Adirondacks area of New York State has directed attention to the similar area of Grenville formation in Ontario and Quebec. Drilling during 1950 and 1951 has proved three deposits in Ontario and one in Quebec, and drilling is proceeding on others. The Bethlehem Steel Corporation has announced its plan for operation of its deposit at Marmora, north of Belleville. Stripping has commenced of the 100 feet or more of limestone that overlies the magnetite. The intention is to produce 400,000 tons a year of magnetic concentrate, which will involve mining nearly one million tons of ore.

The owners of the other three deposits have made no public announcement. The tonnage of ore proven in each case is probably somewhat smaller than at Marmora, but a similar tonnage of magnetic concentrate is planned for each.

As the search for ore of concentrating grade in this large district is only well begun, it seems likely that other deposits will be found.

Steep Rock

In the bed of the former Steep Rock lake two mines are now proven and under development and a third is proven but no plan for development has been announced. There is evidence of other possible orebodies, and drilling is proceeding on some of them.

The Errington open pit has produced about 7 million tons of high-grade hematite since 1945. The development of an underground mine is well under way, with three levels at 200-foot intervals. The ore in this block, 600 feet deep, might be considered as proven ore. The Errington reserve, on this basis, is over 20 million tons, sufficient for about twenty years of operation. The mine is designed for a depth eventually of 2,500 feet, that is six more levels. Thus the probable ore can be taken as 40 million tons. There is no evidence that the ore will change in character or volume below 2,500 feet depth.

The Hogarth mine is now being stripped for production, first as an open pit and later as an underground mine. Its area is about double that of the Errington, so its reserve can be considered double, namely 50 million tons proven ore and 100 million tons probable ore.

The Inland Steel deposit is stated to be as large as the Hogarth, possibly larger; but no official announcement has been made as to the tonnage proven. Meantime, it might be reasonable to assume 50 million tons proven by drilling and 100 million tons probable ore.

Pacific Coast

So little work has been done on the magnetite deposits of the Pacific coast that an estimate of tonnage is bound to minimize the amount actually available. There are probably half-a-dozen deposits at least, each containing a million tons or more. This tonnage will ensure a supply for a 100-ton smelter at Vancouver for an indefinite period. If, as seems likely, the smelter is enlarged in a few years to 200 tons daily or more, the ore supply will still be ample. To mine and concentrate 70,000 tons, or 140,000 tons, of ore a year is not economical, so the present export of ore to Japan in moderate amount can be considered a benefit to the British Columbia smelter, because it permits mining on a scale that can give moderate costs. No doubt the provincial authorities will make sure that no excessive amount of magnetite is exported that might jeopardize the future supply of the local smelter.

Summary

The following points may serve to summarize the iron ore supply of Canada:

(1) In the east, there is an adequate supply for Canadian furnaces for an indefinite period.

(2) For the prairie provinces and on the Pacific coast there is a moderate supply of ore for local needs, sufficient for many years if conserved.

(3) In the east there is a large surplus of ore, beyond Canadian needs, which is available for Europe and the United States.

(4) The interchange of Canadian and United States ores in Ontario is likely to continue.

(5) There will be a greatly augmented iron ore supply when economical processes become available for extraction of iron from sulphide ores.

The Past, Present, and Future of the Canadian Steel Industry*

By J. CONVEY† and S. L. GERTSMAN§

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THE EXPANSION of steel capacity is seldom spectacular. It may be for this reason that the remarkable progress made in Canada has not been generally recognized. In 1950, Canada's output of steel ingots amounted to 3,270,000 tons (1), which is a 118 per cent increase over the tonnage produced in 1939. It can therefore be readily seen that Canada's growth curve for steel production is climbing at a rate paralleling her growth as a nation.

It is by no means an exaggeration of its importance to consider the steel industry as the cornerstone in the economic structure of Canada's manufacturing activities. From the humble diaper pin to the giant members of massive bridges, fortifications, and other steel structures, steel accompanies us as a necessary and inescapable companion in our journey through life. The *Age of Steel* has a literal meaning, and refinements in the manufacture of the products of steel are adapting it to more and more varied uses as time passes.

The integrated iron and steel works of Canada fall into two natural geographical divisions. One has its base on the Great Lakes and draws its supply of coal and some of its ore from the United States. In this division lie the works of the Algoma Steel Corporation, at Sault Ste. Marie, at the junction point of lakes Huron, Superior, and Michigan; and the works of The Steel Company of Canada, at Hamilton, on lake Ontario.

The other division is in eastern Canada, in the Province of Nova Scotia, and is entirely self-dependent, having ample supplies of coal in that Province and of ore in the neighbouring Province of Newfoundland. Here we find The Dominion Steel and Coal Corporation.

In order to appreciate the build-up in steel making facilities, it is of interest to trace the history of the industry in Canada. In 1736, La Compagnie des Forges was empowered by Louis XV of France to erect an iron works in the Seigneurie of St. Maurice, near Three Rivers, Quebec. Limonitic ore had been discovered there as early as 1667. They cast cannon, mortars, iron stoves, kettles, etc. They tried to make steel but were not too successful. The blast furnace operated by this Company was the first on the American continent.

The Radnor Forges plant, ten miles north of Three Rivers, was established in 1860. It was extensive, consisting of a blast furnace, forge, and large rolling mill. The mill was destroyed by fire some years later and was not rebuilt. The ore was bog iron, partly brought in by farmers, but mainly dredged by the Company from Lac Tortue. The output was about 4 tons of iron a day. In 1892, new owners took over the property and erected a furnace with 25 tons daily capacity. This remained in operation until 1911, when it could no longer compete with modern plants. The iron made at Radnor Forges was much in demand for car wheels.

The first blast furnace in Ontario was erected at Lyndhurst, in Leeds county, at the falls of the Gananoque river. This was in 1800. After two years it was closed down. The next attempt was made at Normandale, in western Ontario, in 1815. This furnace, using bog iron as ore and charcoal as fuel, was operated by various owners until 1847, when depletion of the ore forced cessation of activities. Attempts to make iron at Marmora, Madoc, Houghton, and Furnace Falls, all in Ontario, were unsuccessful.

It is interesting at this time to compare the production of these early furnaces with that of a modern iron and steel plant. At Radnor Forges, in 1895, for example, 600 employees produced 7,900 tons of iron, or slightly more than 13 tons per man-year. At Sault Ste. Marie, the Algoma Steel Corporation, with approximately 6,000 employees, operates a plant with a capacity of 1,000,000 tons of pig iron and 1,000,000 tons of steel per year, or about 330 tons of iron and steel per man-year.

Probably the most ambitious of the early efforts to develop an iron industry in Canada was that of the Acadia Mining Company, at Londonderry, Nova Scotia, who produced iron of high quality from ore found in the Cobequid hills. The original furnace installed at the inception of the plant (about 1850) continued 'on blast' until 1874, and its total production is estimated at about 45,000 tons of pig iron. In 1873 (2), a group of English financiers took over the assets of the Company and installed what was then the most modern equipment for the production of steel. One large blast furnace, designed for making pig iron with coke, was built, and equipment was installed for the production of steel by the then newly invented Siemens 'open hearth' process. The Company was apparently originally organized with the main intention of installing a rail mill for the production of rails to be supplied to the Canadian Pacific Railway Company, which was then considering the construction of its lines into Nova Scotia. Being the only steel plant of any consequence, at the time, in Canada, the prospects for success of the venture seemed decidedly favourable. However, the postponement of the proposed construction by the C.P.R. had a serious effect on the fortunes of the

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(1) For references, see end of paper.

steel plant; and, in addition, the experiments made with the Siemens process did not succeed as expected. After several reorganizations and changes in the nature of the operation, the plant was finally abandoned in 1913.

After the middle of the nineteenth century, two vitally important inventions revolutionized the steel industry. One was the invention, in 1856, of the *Bessemer Process,* by Henry Bessemer of England; the other the *Open Hearth Process,* invented by the Martin Brothers of France in 1864. These inventions marked the closing of what might be called the 'Iron Era', and the birth of an era in which the production and consumption of steel was to reach gigantic proportions.

It will be seen that the effort to establish an iron industry in Canada was slow in progress, but persistent, despite many failures and the expenditure of much money with varying degrees of success. There was some definite advance in the industry in the early years following the introduction by the Government, in 1883, of a bounty to assist the production of pig iron. The granting of a bounty was continued until 1911, covering a period of twenty-eight years, and proved to be a needed stimulant for the successful promotion of the industry in Canada.

In the early days, charcoal was used for the reduction of the iron ore. As the scope of operations increased, however, this was found too expensive and impractical, as blast furnaces were being built of increasing size and capacity, and it had been discovered that coal could be converted into coke, an excellent reducing medium, by treating it in the same manner as wood had been treated for conversion into charcoal. Coke is a much purer form of carbon than the coal from which it is made and has an advantage over charcoal of a much greater mechanical strength, as a consequence of which it is better able to carry the weight of the charges, especially in blast furnaces of the size that are built today.

When the manufacture of pig iron commenced in Canada as an established industry, no coke was made in this country and, in fact, very little was made anywhere in North America, except at the coal mines. The introduction, however, of the modern by-products recovery coke oven brought about a complete change and now involves an integrated plant with units for the production of coke, pig iron, and steel.

In Canada there are four such plants, namely, those of The Dominion Steel and Coal Corporation, in Nova Scotia, The Algoma Steel Corporation, at Sault Ste. Marie, Ontario, The Steel Company of Canada, at Hamilton, Ontario, and, also at Hamilton, the more recent plant of the Dominion Foundries and Steel Company, Limited. The annual capacity of these plants, computed for the year 1950, was as follows:

Dominion Steel and Coal Corporation:

15 open hearths and one electric furnace produced approximately 790,000 tons of steel.

Algoma Steel Corporation:

12 open hearths produced 1,000,000 tons of steel.

Steel Company of Canada:

13 open hearths and one electric furnace produced 1,100,000 tons of steel.

Dominion Foundries and Steel Company:

4 open hearths and 5 electric furnaces produced 350,000 tons of steel.

Tests on the use of oxygen in the open hearth and blast furnaces were first carried out in Belgium in 1913. After this date, some further work was done in Germany, Russia, France, and Japan. It was at The Steel Company of Canada's plant, however, that the procedure was brought into sharp focus on this continent and that some of the most useful work was carried out. On Sunday, June 16th, 1946, the Company ran the first real test of applying oxygen to the flame of an oil-fired open hearth furnace (3). This was the beginning of what was to be a valuable development in steel melting practice. Hot metal was added thirty-five minutes after scrap was all charged. Oxygen was turned on five minutes after charge began and continued for four hours and forty-five minutes, then turned off. The heat was tapped for a 0·50/0.55 carbon tie plate steel three hours and twenty minutes after the oxygen was turned off. Total time of the 175-ton heat from charge to tap was eight hours and ten minutes. The usual time on a heat of this type, using the regular fuel, had been twelve hours.

After a series of trials of this nature (4) it soon became apparent that the use of oxygen in open-hearth combustions could hardly be justified by the saving in fuel or the small increase in yield due to faster melting and lower oxidation of the scrap. The deciding factor was the increase in production rate obtainable, and the relative importance of such an increase to a particular operation.

The value of an hour of open hearth time, which is the most important item to be balanced against the cost of the oxygen used, varies widely from plant to plant, depending on the percentage of steelmaking facilities being used and the balance between steelmaking and rolling mill capacity.

Operating at full production, where steelmaking capacity is at a premium, a considerably higher emphasis can be placed on the rate of production than when furnaces are idle for lack of orders. To obtain full benefit of oxygen used for combustion, it is necessary to provide for fast charging of the scrap, since fast melting pre-supposes fast charging and adequate scrap preparation and transfer facilities.

These preliminary observations, based on the trials carried out by The Steel Company of Canada, have proved to be a milestone in the use of oxygen by the steel industry, both in the United States and Canada. A large number of companies started to conduct their own tests on the use of oxygen at various points in the melting procedure. Today, oxygen has made a place for itself in the steel industry. It is used in the bath through a lance in carbon reduction. This use of oxygen has been of particular value to the operator who makes a lot of 0.03 and 0.05 maximum carbon heats. The use of oxygen in the flame is being applied in production mainly in small cold-charge furnaces or where a high scrap/hot metal ratio exists.

Another steel producer who has recently installed blast furnace capacity is Dominion Foundries and Steel. For forty years this Company had produced in Hamilton, Ontario, castings and rolled products such as boiler plate and tin plate from scrap. The output began to exceed the readily available supply of scrap. The Company blew in an 800-ton blast furnace in the fall of 1951. The coal and ore are purchased from the open market.

Atlas Steels Limited, at Welland, Ontario, was organized primarily for the production of die steels. It has become an important producer of electric furnace specialty steels such as hollow drill-steel, stainless steel, and other high-alloy materials.

At Winnipeg, a small rolling mill is operating successfully on scrap.

There are a number of smaller steel plants that subsist on scrap. One of them, in Vancouver, is close to the limit of the available scrap supply. Further expansion will require the production of iron from ore. Some metallurgical coal and magnetite ore are conveniently available along the coast for this purpose. The market at the present time, and probably in the immediate future, is too small to operate a large blast furnace economically. Since electric power is available, it can be visualized that this section of Canada could be supplied with pig iron by the use of electric smelting. The Tysland Hole furnace (5) is, at the present time, the most satisfactory furnace for electric smelting of iron ore.

Since the requirements on the West coast can be expected to be less than 600 tons per day in the immediate future, which is the minimum for a large blast furnace, this type of furnace would suit the location admirably, since it produces about 100 tons of pig iron per day.

It is quite probable that the first continuous casting of steel billets in Canada will be carried out on the West coast. The Vancouver Steel Company, which is the ingot supplier for the Vancouver rolling mills and the Pacific Bolt Company, has already shown an interest in the continuous steel casting process and is investigating its possibilities.

What are the possibilities in the future of a primary steel industry in the prairies? It is now well established that large oil and gas reserves are available in Alberta. The fields in this region are being developed very rapidly. The result has been that the industrial potential of the prairies is now being recognized for the first time. The establishment of new industries is proceeding at a constantly accelerating pace. With the advent of new industry, an increase in the population of this region should inevitably follow and the demand for iron and steel products should increase.

It is well known that the most important factor in deciding whether or not an iron smelting industry should be established in a certain location is the size of the market and the area which can be reached at a competitive price.

For purely economic reasons, a blast furnace smelting plant must be located close to the market for its product and close to the source of fuel rather than close to the iron ore. One of the reasons for this is that it is cheaper to bring the iron ore, at a low 'commodity' freight rate, to the coal deposit and the market for the finished product, than the reverse. Finished iron and steel have much higher freight charges.

Should a smelter be desirable in the prairies, where would its location be? Alberta would be the most probable location, for a number of reasons. First, its population is growing rapidly at the present time and should continue to grow for some time. It has been previously stated that the area which can be reached at a competitive price is an important factor in locating a smelter. The Algoma Steel Corporation, at Sault Ste. Marie, can service the eastern part of the prairies too readily, from a competitive standpoint, to encourage the building of a smelter much closer to the East than Alberta.

What is the raw material situation in this area? Good coking coal is available in several localities in the Rockies and foothills. The excellent coke made by the Crow's Nest Pass Coal Company at Fernie, B.C., and the beehive coke of the International Coal and Coke Company, at Coleman, Alberta, are available. Similar coking coals occur at intervals for some hundreds of miles northwest and in the foothills.

Limestone of a quality suitable for fluxing can be obtained in many localities throughout Alberta. It is unlikely that any site selected for a smelter and rolling mills would be far distant from the limestone needed.

Iron ore for a small operation is now in sight. The Consolidated Mining and Smelting Company of Canada, Limited, recently announced plans for the production of phosphate fertilizer. The sulphuric acid to make this product will come from burning the pyrrhotite tailings from the Company's Sullivan mine. The mineral pyrrhotite contains 40 per cent sulphur and 60 per cent iron. The residue after the sulphur is burned off is an oxide of iron which, when suitably sintered or made into pellets, is a high-grade ore of iron. At the outset, the plant will produce only about 200 tons per day. As the industrialization of the prairies increases, no doubt this plant will be enlarged and more iron oxide will be available. Sulphur required for other chemical plants will release additional quantities of iron oxide. Since 2,000 tons of pyrrhotite tailings are produced daily, and since there is an accumulation of many millions of tons, it can be seen that sufficient ore is available for a smelter for a considerable period of operation.

There are some undeveloped deposits of hematite at Kitchener, B.C. Until further drilling is undertaken, however, they may be regarded as small. At Burmis, near Blairmore, Alberta, there are beds of beach sands which contain magnetite. Magnetic concentration might produce a small but useful tonnage of ore. All of the above indicates the possibility that, in the future, there may be a small primary steel producer in Alberta.

What does the future hold in store for the steel industry in the two great industrial Provinces of Ontario and Quebec? In 1939, Canada had an ingot capacity of 200 lb. *per capita*. This jumped to 480 lb. *per capita* in 1950: and by 1953 our *per capita* capacity will be well over 500 lb. In the United States, the present capacity, on this basis, is about 775 lb. It can be seen that, in Canada, the industry can expand considerably before we reach the *per capita* ingot capacity of our neighbours to the South. Furthermore, should our population increase rapidly due to a change in immigration policy, then our steel industry will likely have to expand that much more. The ore required for such an expansion is available at Steep Rock and Michipicoten, in Ontario, at Wabana in Newfoundland, and in the Quebec-Labrador region. It can be expected that our present steel mills in Ontario would be foremost in any expansion of steel capacity in Canada.

In connection with the use of Labrador ore by the Steel Company of Canada, there is at the present time an economic factor involved. The ore will be shipped by rail to Seven Islands where it will be loaded onto boats. These ships can then ply the St. Lawrence. However, only 2,500-3,000-ton ships can be accommodated through the Lachine canal. Therefore the size of the ore boats is limited to this tonnage figure if shipment is to be by boat to Hamilton. The vessels now transporting ore from the Lake Superior regions to Hamilton bring in cargoes of 15,000 to 20,000 tons. The Company would consequently have to handle a greater number of boats from Labrador than they are presently looking after. This would present the problem of adequate docking and unloading facilities. In the future, however, should the St. Lawrence Deep Sea Waterways be built, larger vessels would be employed for ore haulage, and the

previously mentioned problem would disappear.

In the Montreal-Three Rivers area of Quebec lies a wealth of electric power. It is estimated that a total of approximately seven million horsepower will be available for use in the near future. With this power potential, the ready market for steel products, and the strategic situation for export, is it not reasonable to anticipate the possibility of an electric smelter in this area? This type of smelter would appear to be ideal, especially since there is no coal available in this area, and an electric smelter uses less than half the amount of coke in the charge than does a blast furnace.

A similar possibility exists along the Labrador ore deposits, however, only if it should be desired, for national defence purposes, to disperse our steel industry. There are in the area several potential sources of hydro-electric power of large size and capable of low-cost development within easy transmission distance. Since it is unlikely this power can ever be transmitted to the shore of the St. Lawrence, a case can be made for using it close to the orebodies.

A further possibility derives from the present continent-wide shortage of scrap, which is likely to persist for some time. The Wiberg-Soderfors sponge-iron furnace may prove suitable for use with the Labrador iron. Sponge iron made by this process is an excellent substitute for scrap in the electric furnace. There is a trend toward increased use of the electric furnace for steelmaking in the United States. Should that trend follow in Canada, the Wiberg process may well come to the fore.

The Labrador ore may help the Dominion Steel and Coal Corporation at Sydney, Nova Scotia, to produce steel cheaper in the future. Dilution of the Wabana ores, which have high phosphorus and silica content, with Labrador ores, may result in lower steelmaking costs. A factor such as this may increase the market for the Company's product by making it more competitive in the future.

Another development which will have an effect on Canada's steel capacity in the future is the large titanium project presently being developed by Kennecott Copper and the New Jersey Zinc Company at Allard lake, Quebec, with electric furnaces at Sorel. This is entirely independent of the Labrador ore project.

As a by-product of this operation, a 1.15 per cent carbon, high-sulphur steel is obtained. By further refining in an electric steel melting furnace, the carbon and sulphur contents are reduced to conform with the specification requirements of standard grade steels. In the future, this can develop into a fully integrated steel plant based on electric power, producing about 2,000 tons of ingots per day.

In regard to the development and use of new steelmaking processes in the future, such as the production of steel direct from the ore, the use of hot-metal no-scrap charge steelmaking processes, faster and more efficient methods of desulphurization, continuous casting, and so on, it will be found that Canada will not lag behind.

This paper has traced the history of the steel industry in Canada from 1736, when a small company erected an iron works near Three Rivers, through to today, when we have four primary steel producers. These account for most of the 3¼ million tons of steel which make Canada the sixth largest steel producing country in the world. If, in prophesying the future possibilities of the Canadian steel industry, we are only half as successful as a previous Canadian, Mr. C. S. Cameron, we will be doing well indeed. Mr. Cameron, in an article written in 1925, stated: "In the region recently added to the Province in the district of Ungava, there are indications of large deposits of minerals, especially of iron ore. This region lies between Hudson bay and the Labrador coast of Baffin bay and Hudson strait, and has not yet been thoroughly explored although it has been traversed by geologists and others capable of reporting the main features of the country correctly and intelligently. Their reports make it certain that, in an area not less than two thousand miles in extent, there are very large deposits of iron ore of high commercial value containing about 50 per cent of metallic iron. Although this region is now beyond the range of practical access, it would appear that the chief value of the deposits will be as a reserve for the supply of ore after more accessible sources have been exhausted". How true this is today, twenty-six years later, when the high-grade Mesabi ores are nearing exhaustion and work has started on opening the Labrador ore deposits.

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Asbestos Floats

*By M. S. BADOLLET**

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Introduction

ASBESTOS FLOATS may be defined briefly as blends of airborne particles of fibrous asbestos and dusts produced during the milling stages and collected by Cottrell precipitators, dust sheds, or bag-houses.

The various grades of asbestos floats on the market contain fibrous asbestos particles ranging in length from microscopic to approximately ¼-inch. The fines or dusts present in the floats are from 40 microns to 2 microns or less in cross-section.

Airborne asbestos particles are of diverse length because they are collected at numerous places in the mill. There are: (1) dusts removed from dry rock storage bins, (2) dusts removed from rock and fiber screens, (3) particles from the tops of collectors, and (4) fines removed by suction from the processing equipment.

Sometimes the airborne particles are kept segregated, so that there is a partial separation by gravity. This is accomplished by a series of dust chambers enclosed in a long, rectangular building. The longest and heaviest particles fall nearest to the air flow entrance, the smallest at a point farthest away and nearest the exit. Sometimes the air from the dust chamber building is passed through a series of bag-houses or electric precipitators to collect the extremely fine particles which normally escape to the atmosphere.

In other instances, the airborne particles by-pass the dust chambers and are passed through rows of bag coulectors.

Regardless of the method of collecting the particles, the milling department must select proper fiber sizes for blending, producing uniform floats to meet the requirements of the trade.

Sometimes one grade of float can be used successfully in many commercial products; in other cases, great care must be taken to produce a float for use in a specific product.

The present total production of all grades of Canadian floats approximates 40,000 tons per year and it is possible that this quantity may be greatly increased by the proposed expansion programmes now going into effect in Canada.

Since floats are valued at $40 per ton at the Canadian mines, a tonnage of 40,000 has a potential sales value of $1,600,000. This division of the asbestos industry is a good business, and it is well worth the concentrated effort of research and development engineers to find new uses, and so expand the market, for all available floats.

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Physical Properties of Floats

According to the Quebec Screen classification, there are two general classes of floats, known as *7RF* and *7TF*.

Samples of various commercial floats were obtained on the market and a series of tests were devised to determine the differences in their physical properties. Some of these tests were similar to those made by many manufacturers who use floats in their products. The results of some of these tests are given in Table I. They do not, however, include the official Quebec Screen test. Manufacturers using floats usually set up their own specifications, based upon requirements for their particular products.

Loose Density

The loose density of floats can be measured by slowly sifting the powder into a can or container of definite dimensions so that the weight per cubic foot can be calculated.

Although in some products the loose density of the floats is not particularly important, in others it is critical, so each float must be considered as an indivdual case.

In the present investigation eleven *7RF* floats were tested and their loose densities were found to range from 14 to 25 pounds per cu. ft. The

Table I

Identification	Loose Density lb./cu. ft.	Material +100 M %	Fines −200 M %	Grit +48 M %	Whiskers +48 M %	Grit −48 M +80 M %	Whiskers −48 M +80 M %	Oil Adsorption c.c./g. fiber
(1) 7RF....	19	3	96	0.04	1.90	0.02	0.64	0.66
(2) 7RF....	20	4	93	0.14	2.50	0.14	0.66	0.74
(3) 7RF....	21	4	93	0.24	2.26	0.12	1.18	0.75
(4) 7RF....	22	4	93	0.16	1.30	0.20	0.94	0.68
(5) 7RF....	22	3	95	0.12	0.80	0.10	0.46	0.65
(6) 7RF....	24	5	90	Tr	1.16	0.04	0.96	0.68
(7) 7RF....	24	5	89	0.06	1.28	0.10	1.12	0.71
(8) 7RF....	20	6	91	0.12	4.12	0.56	1.10	0.66
(9) 7RF....	25	7	84	0.24	0.70	0.46	1.30	0.65
(10) 7RF....	23	7	88	0.46	2.38	0.54	0.92	0.65
(11) 7RF....	14	1	96	0.04	0.50	0.02	0.84	1.05
(12) 7TF....	23	2	96	Tr	0.30	Tr	0.50	0.75
(13) 7TF....	23	2	95	0.06	0.60	0.06	0.70	0.65
(14) 7TF....	26	3	91	0.00	0.10	Tr	0.16	0.68

heaviest is No. 9 and the lightest No. 11. Nos. 1 to 5, inclusive, and No. 8, all with loose densities between 19 and 22 pounds per cubic foot, may be considered as falling within the same range, as also may Nos. 6, 7, 9, and 10, which are slightly heavier (23 to 25). Float No. 11 is in a class by itself and is light in weight as well as bulky (loose density 14). If density were the only physical property that had to be considered, this record would enable the manufacturer to select the proper one for his purpose.

The three *TF* floats included in the Table differ from the *RF* floats in degree of particle fineness. Nos. 12 and 13 have identical loose density (23) while No. 14 is heavier (26).

Classification by Wet Washing

Fibre and Grit +100 Mesh

A classification of the particles can be obtained by a wet-washing method which determines the amounts of fiber and grit plus-100-mesh and of fines minus-200-mesh. This information is particularly useful if the floats are to be used in molding, extrusion, or spraying.

The *RF* float No. 11 contains only 1 per cent of material larger than 100 mesh. In the other *RF* floats, the content ranges from 3 to 7 per cent, indicating the presence of material that is sometimes referred to as 'whiskers'. In the plastics industry, particles plus 100 mesh may be helpful in producing impact strength in the product.

The three *TF* floats contain almost identical amounts of plus-100-mesh material. In two of them the percentage is lower than in the first ten *RF floats.*

Fines —200 Mesh

The fines minus-200-mesh in the *RF* floats range from 84 to 96 per cent, and in the *TF* floats from 91 to 96 per cent.

The material between plus 100 mesh (fiber and grit) and minus 200 mesh (fines) would be collected on the 200 mesh screen. Its amount can be obtained by adding the plus 100 mesh and minus 200 mesh values shown in the Table and subtracting the sum from 100.

This test, although useful, does not distinguish between, or give the relative amounts of, 'whiskers' (asbestos fibers) and particles of grit that might be present in the floats.

Grit +48 Mesh

In many cases, the grit particles in the floats are of very diverse size. Usually, they are small pieces of serpentine rock and magnetite, or small bundles of asbestos fiber. Complete classification, therefore, requires a further breakdown to determine the quantities of these objectionable particles present in the floats.

This test is made by water elutriation and by careful separation of the grit from 'whiskers' or fiber bundles.

The amount of grit plus-48-mesh in the eleven *7RF* floats ranges from a 'trace' to 0.46 per cent. The percentage is lowest in samples. Nos. 1, 6, and 11 and highest in No. 10. Although these quantities of grit seem small, they could cause considerable trouble in a manufacturing process by plugging extrusion apparatus or scoring the walls of expensive molds.

The three *TF* floats are almost free from grit plus-48-mesh.

'Whiskers' +48 Mesh

'Whiskers' are short asbestos fibers that may be valuable or objectionable, depending upon the use for which the product is required.

The quantity of whiskers in the eleven *RF* floats ranges from 0.50 per cent (in No. 11) to 4.12 per cent (in No. 8). In the *TF* floats, the range is from 0.10 per cent (No. 14) to 0.60 per cent (No. 13).

Grit —48 Mesh +80 Mesh

In some special products, the presence of grit smaller than 48 mesh, and even smaller than plus 80 mesh, is considered objectionable. Therefore, a further separation is made to determine the quantities of this grit that may be present in these floats.

As shown in Table I, the range for the elveen *RF* floats tested is from 0.02 per cent (Nos. 1 and 11) to 0.56 per cent (No. 8).

Two of the *TF* floats are free from this size of grit and the third (No. 13) contains only 0.06 per cent.

'Whiskers' —48 Mesh +80 Mesh

The presence of whiskers in the sizes of minus 48 mesh and plus 80 mesh is also objectionable because they affect the surface of certain products.

The eleven *RF* floats tested contain whiskers of this size ranging from 0.46 per cent (No. 5) to 1.30 per cent (No. 9). Corresponding figures for the *TF* floats are a low of 0.16 per cent (No. 14) and a high of 0.70 per cent (No. 13).

In the manufacture of plastics, the presence of whiskers results in non-uniform pourability of the molding powders. This necessitates a re-screening operation so that uniformly coated powders will properly fill the molds. In many cases, the whiskers and crudy fiber bundles are not thoroughly covered by the organic plastic material and this increases the water absorption properties of the plastic material, which may result in the formation of blisters.

Oil-Adsorptive Capacity

The capacity of a float to adsorb oil is important because it gives an indication of ability to hold an organic liquid when manufacturing a plastic or caulking compound.

This test, made by manufacturers, consists of titrating a given weight of float with oil until its adsorptive capacity is reached. In practice, the adsorptive capacity is considered to have been reached when oil can be released from a ball of the thoroughly mixed float and oil by applying a small amount of pressure against it. The results of this test are expressed in terms of cubic centimeters of oil per gram of float.

For the eleven *RF* floats tested, the oil adsorptive capacity ranges from 0.65 to 1.05 cubic centimeters per gram. The value for No. 11 (1.05 c.c./gm.) is much higher than that for any of the others, which all fall between 0.65 and 0.68 c.c./gm. except Nos. 2, 3, and 7 (0.71 to 0.75 c.c./gm.).

Two of the *TF* floats, Nos. 13 and 14, have closely similar absorptive capacity, 0.65 and 0.68 c.c./gm., respectively. No. 12 gave a higher value (0.75 c.c./gm).

Miscellaneous Physical Tests

Numerous other tests, of importance to the manufacturer of certain asbestos products, are made on floats. Data for these are given in Table II and are summarized in the following paragraphs.

Surface Area

Study of the surface area of each float was made to obtain data on its covering power as a filler. In this test, the surface area is measured by the air permeability method by means of the Bowen apparatus, fol-

TABLE II

Identification	Surface Area	Density lb./cu. ft.		Compressibility* % Thickness at 200 Compress.		Springback, % Thickness at Max. Compress.		Surf. Cond. of Pressed Specimen	Ionizable Salts, Micromhos/ cm/10 volts	Dry Bulk, Vol. of 100 g. cc.	Water-Soluble		Magnetic Rating
											Solids %	Chlorine as Cl %	
	cm^2/gm.	2,000 p.s.i.	3,000 p.s.i.	2,000 p.s.i.	3,000 p.s.i.	2,000 p.s.i.	3,000 p.s.i.						
(1) 7RF	14,800	89.9	94.7	29	32	2.5	2.8	Sm'l whiskers & bundles	219	350	0.16	0.017	4.7
(2) 7RF	11,300	93.6	97.0	28	32	3.1	3.7	"	218	335	0.17	0.024	6.3
(3) 7RF	13,000	93.5	99.2	28	32	3.2	3.3	"	222	340	0.18	0.029	6.1
(4) 7RF	11,800	95.5	99.1	29	32	3.3	3.4	Grit & few whis. & bund.	206	305	0.16	0.025	6.0
(5) 7RF	13,000	95.0	98.2	28	31	2.6	3.4	Sm'l amt. of grit & whis.	212	300	0.17	0.022	5.7
(6) 7RF	14,000	100.0	105.4	28	32	3.8	4.0	Good, smooth	379	320	0.36	0.15	4.1
(7) 7RF	13,900	100.8	105.2	28	32	3.4	4.0	"	393	295	0.38	0.16	5.1
(8) 7RF	12,500	102.7	107.7	28	32	3.6	4.2	Grit mostly	421	360	0.26	0.15	5.1
(9) 7RF	7,500	99.6	105.8	27	31	3.4	3.9	"	190	255	0.12	0.0049	5.9
(10) 7RF	8,600	101.2	105.8	28	31	3.0	3.4	"	243	285	0.14	0.0038	9.6
(11) 7RF	20,800	93.6	99.2	35	39	5.8	6.7	Good surface	260	475	0.15	0.0015	3.4
(12) 7TF	15,400	95.0	100.0	27	31	3.3	3.7	Good surface	221	295	0.15	0.022	5.0
(13) 7TF	11,300	94.8	99.4	27	30	3.5	3.3	Large grit	176	290	0.14	0.0034	4.0
(14) 7TF	15,600	96.0	99.5	27	29	3.3	3.3	Good surface	396	275	0.36	0.16	3.9

*Measurements start at 200 p.s.i. as zero point.

lowing the Lea and Nurse technique (1).

The eleven *RF* floats tested show a wide range in surface area, from 7,500 cm^2/gm. (No. 9) to 20,800 cm^2/gm. (No. 11). In other words, float No. 9 has the lowest, and float No. 11 has the greatest, covering power as a filler. In plastics, the surface area of the floats governs the amount of filler that can be used successfully prior to pressing in a given mold.

Two of the three *TF* floats (Nos. 12 and 14) have the same surface area, approximately 15,500 cm^2/gm. For No. 13 the value is 11,300 cm^2/gm. The high value of 15,500 for Nos. 12 and 14 indicates the presence in these floats of a large percentage of fines.

Density at 2,000 and 3,000 p.s.i.

Ten-gram samples of each float were subjected to pressures of (1) 2,000 p.s.i. and (2) 3,000 p.s.i. and their densities calculated in pounds per cubic foot. These pressures were selected because they correspond to the range of pressures used in the plastics-molding industry, and the data obtained are helpful in selecting the proper asbestos filler for molding powders.

At 2,000 p.s.i., the densities range from 89.9 lb./cu ft. (No. 1) to 102.7 lb./cu ft. (No. 8). Several of the samples have substantially identical density, but for the group as a whole the differences in density are such as would influence the selection of the filler to be used for certain products.

At 3,000 p.s.i., the densities range from 94.7 lb./cu. ft. (No 1) to 107.7 lb./cu. ft. (No 8). The increase in density for the additional 1,000 p.s.i. pressure ranges from 3.2 to 6.2 lb./cu. ft.

Compressibility of Floats at 2,000 and 3,000 p.s.i.

In this test, a 10-gram sample of the float is compressed at 200 p.s.i. and its thickness measured. It is then compressed at (1) 2,000 p.s.i. and (2) 3,000 p.s.i. and its thickness again measured. The reduction in thickness, expressed as a percentage, gives a measure of the compressibility for each float. Thirteen of the fourteen floats tested show about the same percentage of compressibility (27 to 29). For No. 11, the value (35) is appreciably higher.

This is an important property and must be taken into consideration when pressing a molding powder to a definite size after the molding cavity has been properly filled. Uniform compressibility is important as a bulky fiber will also exhibit poor dimensions after pressing.

Springback After Release of Pressures

The amount of 'springback' of a molding powder is another important property considered by plastics manufacturers. If the springback is too great, it will be difficult to obtain the proper dimensions for a molded product. Too much emphasis cannot be placed upon this property.

All fourteen floats were tested for springback at pressures of both 2.000 and 3,000 p.s.i. As will be noted from the data in Table II, the springback factor is uniformly low for all of them except No. 11 which, also, is the sample with the highest percentage of compressibility.

To use float No. 11, with a springback of 5.8 and 6.7 per cent, allowance must be made for its compressibility or the plastics manufacturer cannot control the dimensions of his product.

Surface Condition of Pressed Blocks

The surface characteristics of pressed blocks of the floats are noted in Table II. The surfaces usually show long 'whiskers', short stubby fiber bundles, and grit, which generally are considered poor properties from the viewpoints of appearance and water adsorption.

A smooth surface is desirable, provided the molded product meets all other specifications of the manufacturer. Five of the floats give good smooth surfaces and probably will be satisfactory for use in molding powders, provided other properties are acceptable.

Ionizable Salts

For the manufacture of some special types of molding powders, it is essential that the asbestos fibre have low electrical conductivity. In this case, some plastics companies make a test on water-soluble material leached from floats. Apparently, all floats contain a certain quantity of water-soluble material. Its conductivity is determined, and the result of the test is expressed in terms of micromhos per centimeter per 10 volts. For the eleven *RF* floats tested, this factor ranges from 190 (No. 9) to 421 (No. 8). For the *TF* floats, Nos. 12 13, and 14, the values are, respectively, 221, 176, and 396.

It is not known at present which values are objectionable, but it appears that a conductance of 300 micromhos or higher per centimeter per 10 volts may cause trouble, with the molded plastic product having a low dielectric strength.

(1) For references, see end of paper.

Dry Bulk

The bulking values of floats are sometimes considered important factors in plastics as well as in dry-wall joint fillers.

By some manufacturers, the bulking value is expressed as the volume, in cubic centimeters, of a given weight, say 100 grams, of the float. Some fillers should have high dry-bulk value while others should not; this depends entirely upon the intended application.

The eleven *RF* floats have dry-bulk values ranging from a low of 255 (No. 9) to a high of 475 (No. 11). For the three *TF* floats, Nos. 12, 13, and 14, the values are, respectively, 295, 290, and 275 — differences that may lie within experimental error.

Water-Soluble Material

All floats were analyzed for water-soluble materials by allowing a sample of each to stand in distilled water for 63 hours and filtering off the clear water. Aliquots of the filtrates were dried and weighed to determine total solids, and these were then analyzed for chlorides.

The amount of water-soluble material in the eleven *RF* floats ranges from 0.12 per cent (No. 9) to 0.38 per cent (No. 7). In the *TF* floats the range is about the same, with a maximum of 0.36 per cent for No. 14.

No good correlation exists between the ionizable salt values and the quantity of water-soluble material.

The percentage of chlorine in the residues also was determined. For the eleven *RF* floats it ranges from 0.0015 to 0.16 per cent, and for the three *TF* floats from 0.0034 to 0.16 per cent.

Magnetic Rating

Since many floats are used as fillers in products employed by the electrical industry, it may be important to learn something about their magnetic ratings. The test, an adaptation of the Mapes (2) test, measures the amount of magnetite present.

Ratings of the eleven *RF* floats range from 3.4 (No. 11) to 9.6 (No. 10). The other nine *RF* floats have ratings between 4.1 to 5.9, with an average of 5.4 The three *TF* floats range from 3.9 to 5.0, averaging 4.3.

The maximum magnetic rating value for a successful float is not known precisely, but it is believed that the manufacturer would prefer the lowest rating available. It is known that the smallest trace of magnetite can be easily detected in a magnetic field and it may result in the rejection of the plastic material.

Use of Asbestos Floats

Asbestos floats have a definite use in such industrial products as automobile body coatings, adhesives, caulking compounds, dry-wall joint filler, lubricating greases, asphalt and cold-water paints, plastics, molding powders, welding rods, insecticides, radiator scaling compound, acoustical plasters, cements, and many others.

Asbestos floats offer to manufacturers the advantages of being a low-cost inorganic filler, which will improve impact strength and provide good workability, good binding qualities, heat resistance, fair acid and alkali resistance, large surface area, and availability in commercial quantities.

A recommendation for the best grade of asbestos floats for a particular product should be made carefully, since each product has its own requirements and specifications.

Automobile Body Undercoating

A float for automobile body coating to be applied by spray gun may require the absence of 'whiskers' and large particles of grit. If the coating is to be applied by brush, however, the presence of whiskers and grit would not be objectionable. Therefore, the selection of a float for this job depends upon the method of application and such other factors as bulk and fluidity when incorporated in a binder.

Adhesives

The use of floats in adhesives depends upon the type and application of the adhesive product. The proper recommendation can be made only after knowing the desired properties of the adhesive and the method of application. The amount of floats in adhesives (3) varies between 9 and 40 per cent of the mix, depending upon the product.

Caulking Compounds

Caulking compounds may be sensitive to changes in grade of floats. For example, a gun grade of caulking compound may require a float relatively free from whiskers and grit, while, for a caulking compound applied by knife, a float containing whiskers and some grit could be used. In general, the use of floats in caulking compounds (3) may range from 4 to 15 per cent by weight of compound.

Wall Joints

The manufacturer of dry-wall joint filler is very careful in his selection of floats. His specifications for asbestos fiber are rigid and he insists upon good bulking value, freedom from grit, good coverage, and no whiskers. Very few natural floats on the market today fulfill these requirements and, therefore, a special float must be made for this job. The amount of floats (3) required for this use may vary between 5 and 20 per cent of the dry mix ready for the market.

Lubricating Greases

Various types of floats have been used in some grades of lubricating greases, but in most greases grit is objectionable. Whiskers may be permitted unless the customer's specification indicates otherwise. The amount of float used in greases varies between 10 and 25 per cent of the product (3).

Paints

Paints, with either asphalt or cold-water base, may require a grit-free and whisker-free float if smooth surface is demanded. Otherwise, some grit and whiskers are allowable. Paints may contain from 5 to 11 per cent of floats.

Plastics and Molding Powders

Floats, with specifications varying from customer to customer, are used extensively for plastics and molding powders.

The general opinion of the plastics manufacturer is that fibrous float structure increases impact strength, allows a wide range for adjusting quantity of floats as filler, furnishes good binder retention and workability, imparts hardness and toughness to the moldings, increases heat and fire resistance, reduces natural shrinkage of resins and binders, reduces warpage and deformation, makes cold-molding possible by controlling the flow under pressure in the molds, allows blending of mineral and organic fillers, imparts good finish, improves the electrical properties, and reduces the cost of moldings.

Molded plastics are carefully checked for specific gravity, flexural strength, tensile strength, compressive strength, impact strength, water absorption, dielectric strength, power factor, shrinkage, heat resistance, visual observations of mold-

ing qualities, and ejection from the mold. It is important, therefore, that the plastics manufacturer select asbestos floats with care so that molding powder and molded product have the desired properties. The asbestos floats used as a filler in these materials generally form from 60 to 80 per cent of the mix (3).

Welding Rods

The welding rod manufacturer desires a float which will produce good coating as a filler, and which lacks whiskers and grit that might clog extrusion equipment. Floats also should be free from any sulphide mineral, or at least should contain less than the 0.01 per cent of sulphur that would produce obnoxious gas during welding. The amount of floats used for coating rods varies between 8 and 10 per cent*.

Insecticides

Floats sometimes are used as filler for insecticides to be dusted or sprayed onto flowers, weeds, and other plants. In this case, the float should be a fine dust so that it will impart large coverage. Floats for this purpose must compete with such other powders as talc and diatomaceous earth. The amount of float in a ready-to-use insecticide is approximately 4 to 5 per cent.

*British Patent 535,355; Canadian Patent 365,792.

Radiator Sealing Compound

Various combinations of materials, including fine asbestos, are used in sealing compounds for automobile radiators. The asbestos forms from 1/ to about 6 per cent by weight of the solids content of the mixture†.

Acoustical Plasters

Based upon the weight of the dry compound, some acoustical plasters contain from 6 to 15 per cent of asbestos floats. In this application, the floats function as a filler to provide bulk and fire resistance. Whiskers and grit are not objectionable unless a spray gun is used.

Cements

Cements and adhesives are considered similar by some authorities, and the amount of floats needed for adhesives would apply also to cements. Depending upon the type of cement and the application, the asbestos floats content ranges from 9 to 40 per cent (3). In many cases, whiskers and grit are permitted.

Conclusions

(1) Fourteen asbestos floats produced in Canada have been examined in tests similar to those made by customers in their own laboratories.

(2) Many of these floats have almost identical characteristics; however, one float, No. 11, differs considerably from all the others and has many unique properties.

†U. S. Patents 2,391,737, 2,315,321.

(3) The type of float required for a particular product depends upon the use for which the product is manufactured, and each manufacturer has his own specifications and methods for testing the float.

(4) Float characteristics such as grit, whiskers, compressibility, springback, and magnetite rating are usually carefully checked by the plastics manufacturer, since these qualities affect his products.

(5) Bulk, surface area, and density values of floats are considered important factors in such products as wall joint fillers, caulking compounds, some pressed products, and acoustical plasters.

(6) The amount of ionizable salts, water-soluble solids, and chlorides in the floats is a matter of importance and frequently is checked by plastics manufacturers interested in improving the electrical properties of their products.

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Permafrost Problems in Oil Development in Northern Canada

By R. A. HEMSTOCK *

(*Annual Western Meeting, Calgary, Alta., October*, 1951)
(*Transactions, Volume LV*, 1952, pp. 180-183)

OIL EXPLORATION in Western Canada has produced outstanding results in the past few years. From such successes as Leduc and Redwater on the western plains, oilmen are now moving north in the ever-widening search for oil. They are reaching new frontiers and meeting new problems presented by this vast hinterland.

One of the least known but most troublesome phenomena which will be encountered is *permafrost*. This term relates to a thickness of soil or other superficial deposit, or even of bed-rock at variable depths beneath the surface, in which a temperature below freezing has continually existed for a long time. This permanently frozen ground affects, in many ways, oil exploration and exploitation. Seismic records are changed, pipe lines cannot easily be ditched into the ground, formation of wax is speeded-up in tubing and pipe lines, roads are difficult to build, and foundations of buildings and even of drilling rigs may require entirely new methods of construction.

Permafrost underlies roughly one-fifth of the entire land surface of the world. It covers the northerly portions of Eurasia and North America. In the area that we are here concerned with, the southern boundary of permafrost extends south of east from about latitude 59°N. on the Pacific coast across the Rocky mountains, through Great Slave and Athabasca lakes, to cape Henrietta Maria, at the junction of Hudson bay and James bay (see Figure 1). Geologists generally agree that permafrost first appeared in the Pleistocene or Ice Age as the result of abnormal ground freezing. In areas where the mean annual temperature is below freezing, the ground has remained frozen. There is some difference of opinion as to the mean annual temperature required to cause aggradation of permafrost. Russian scientists claim that between 21°F. and 27°F. is required and that a mean temperature of 30°F. will support permafrost. It is of general interest to note that the Russians have also put forth a theory that there should be a geographical relationship between the areal extent of the Pleistocene glaciation and the present thickness of permafrost. However, there is some evidence that very thick permafrost sections occur where there was no glaciation.

Thickness of permafrost varies according to location, from a few feet in the south to over 1,200 feet in the extreme north. At Norman Wells, a thickness of 140 to 200 feet has been recorded. Drilling in Alaska has indicated permafrost to a depth of about 1,200 feet.

The swamps and muskegs of northern Alberta are already well known to oil operators in that area. As one goes farther and farther north, the bush thins out—low-lying areas still predominate, but frost will be found all year round at a depth of a few feet. In general, all exploration for oil and gas in the far north will encounter this sort of terrain—low-lying ground with only a few well-drained ridges running through the country.

Typically, in these areas, there is, at the surface, an 'active' layer, one to five feet thick, in which the soil is subject to seasonal freeze and thaw and from which such troubles as frost heaves, frost mounds, and icings originate. Below this we find permafrost to various depths, succeeded by normal unfrozen material showing the usual temperature gradient. It is in such surroundings that permafrost will first affect the operations of those seeking oil. In the summer, the first trip with tractors or other equipment will be made with little trouble but, as the moss and vegetation is disturbed, the ice and soil below thaws out, resulting in a hopeless sea of mud.

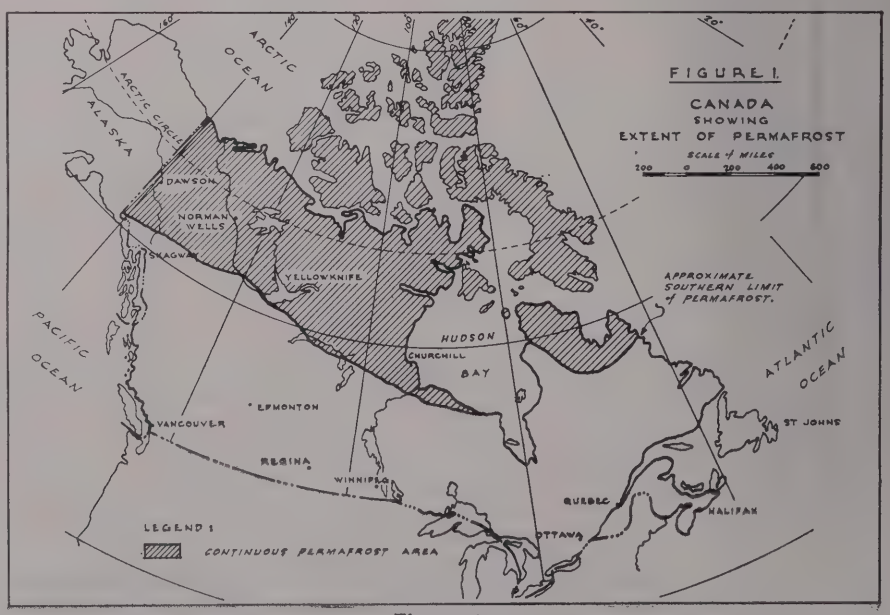

Figure 1.

* Producing Department, Western Division, Imperial Oil Limited.

As a consequence, road building and airport construction are the first requisites of any programme for development of oil resources in northern areas. The presence of permafrost requires that special methods of construction be employed if satisfactory results are to be obtained. At Norman Wells, experience showed that the best results were obtained when moss and scrub from both sides of the road were pushed over undisturbed original moss to the centre of the graded road-bed. This material was then packed and over it was placed a filled sub-grade, either from the ditches or from borrow pits (see Figures 2, 3 and 4). This method is essentially the same as outlined by Muller*; it allows for good drainage, and also causes a rise in the permafrost level in the road-bed. In muskeg areas, all stripping of moss and vegetation and subsequent piling in the road-bed should be done in the spring, after the snow is gone, so that machinery can move easily and efficiently on the still frozen ground. Although this sort of construction has been used for airports in the North, it is generally much cheaper to locate the airport on a suitable area of high, dry ground, even though this may be at some distance from the road.

In all construction on permafrost, drainage is of great importance. The natural growth of moss and scrub holds moisture and keeps the permafrost level very near the surface. In any operation in which this vegetation cover is broken or — as in many cases — is cleared away entirely, the resulting lowering of the permafrost level releases large quantities of water which will remain to hamper operations unless there is adequate drainage. If possible, the building area should be cleared at least two years before it is to be used, and ditches should be provided to carry away excess water. This lowers and stabilizes the frost line and allows for easy work.

In planning construction of buildings in permafrost areas, we find that there are two general methods which may be used. These are known as the 'active' and the 'passive' methods. Where permafrost is thin and sporadic and ground temperatures are above 29.5°F., the frost generally should be eliminated by thawing before construction is started. This is the 'active' method. Where ground temperatures are below 25°F., the frozen ground conditions are left undisturbed, or more insulation is added so that any new thermal regime will not cause thawing of the ground; this is the 'passive' method. Where ground temperatures are between those mentioned, either method may be employed. However, the active method should be used only where the properties of the ground in the thawed state are such that satisfactory foundations can be designed for the soil in its new state. Elimination of permafrost is not generally feasible if it persists to depths of over 40 feet.

The experience of the Russians in Siberia, and lately of ourselves in the Northwest Territories and Yukon, emphasize the fact that an extensive preliminary investigation of permafrost is most important if long-range projects are to be successfully established in these far

Figure 2.—Sub-grade of a road.

Moss and scrub from sides of road have been pushed in to form an insulating base.

* MULLER, S. W., *Permafrost;* J. W. Edwards, Inc., Ann Arbor, Michigan.

Figure 3.—Finished road.

Grade has been hauled in to cover the sub-grade.

Figure 6.—Failure of a power line.

In this line, poles were heaved out of ground by frost action after only two seasons.

northern districts. In normal construction in the south, we have often found that a lack of preliminary investigation has resulted in great losses of time and money. Unless proper precautions are taken, these losses will be many times greater in the north, where permafrost prevails.

Three main factors must be checked before construction of buildings is commenced in new locations. They are:

(1) Texture and structure of the ground
(2) Temperature of ground and air
(3) Hydrology of the ground

A study of these factors, together with a knowledge of the requirements of the structures involved, will enable the engineer to choose the most practical foundations. Commonly, footings for light, temporary buildings in the North are simply log or timber pads, or posts on pads. Larger, more permanent, structures are best founded on piles properly anchored deep in permafrost, with the original ground surface well insulated to prevent thawing. Piles may be wood, steel, or concrete. As a general rule, it has been found that, in order to provide for proper anchorage of piles, they should be driven to a depth in permafrost equal to 2½ times the depth of the 'active' layer. Generally, holes are steam-jetted into the frozen ground and the piles are driven firmly into the thawed holes (see Figure 5).

So many factors enter into a study of permafrost, and its effects are so widespread, that it is impossible adequately to cover them here. Therefore, only a few of the more startling effects on oil operations will be mentioned.

In Alaska, where frost may be found at depths in excess of over 1,000 feet, seismologists have found that their records are greatly affected by any irregularities in the frost line. Tests indicate that wave velocities are nearly doubled where the ground is frozen. Moreover, the depth of permafrost varies greatly from place to place. For example, the lower frost line is much closer to the surface under old lakebeds than elsewhere. As a consequence, the correct interpretation of seismic records is dependent in very large measure on a knowledge of permafrost conditions. No information is available regarding the effect of permafrost where it persists to only a few hundred feet in depth.

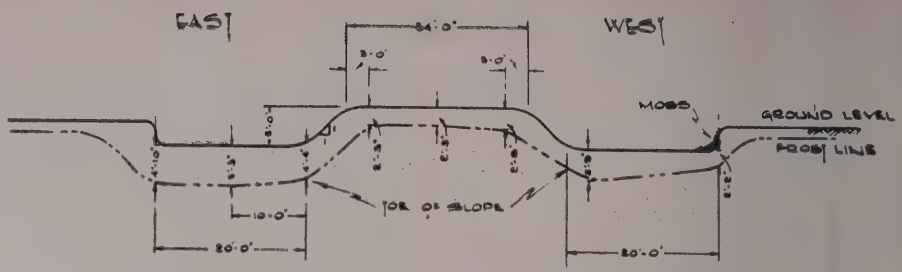

Figure 4.—Frost line on road running N.-S. in a muskeg area.

Frost line in general area is about 16 inches below surface. The resulting desirable frost line in the roadbed is indicated in the Figure.

Figure 5.—Steel piling under a building.

The moss surface has been left in place to help maintain the frost level.

At Norman Wells, where oil has been produced for many years within about eighty miles of the Arctic circle, some difficulty has been found in getting good cement jobs at depths where the ground temperature is below freezing. In general, this difficulty does not arise in deeper holes in which mud has been circulated for some time, with consequent warming of the ground around the borehole. Failing this, some method of heating the cement is required to give it time to set properly. This low temperature at depth also speeds-up wax deposition within tubing well heads and flow-lines. In such cases, the wax must be removed by heating or by mechanical means, as with scrapers.

To date, pipe lines in nearly all these far northern operations have been laid on the surface. The 660-mile Canol line of the last war was on the surface from Norman Wells to Whitehorse. The operation of these surface lines is so unsatisfactory that it is recommended that all future lines be buried. Since the the 'active' layer is seasonable and frozen ground cannot be ditched, lines should be covered wherever possible by pushing moss over them to protect them from the great variations in temperature and the extreme cold in winter.

The construction of water and sewer lines to service northern communities also presents a unique problem. If these lines are ditched to grade, some source heat must be provided to prevent them from freezing. The cost of installations of this sort is further increased by the necessity of ditching frozen ground. Another method that has found favour is to install all lines in a well-insulated box resting on piles above ground. This method allows for easy repair of lines and is flexible and relatively cheap. Steam from a central heating plant may be used as a source of heat.

A reliable source of good water is one of the first concerns of those establishing themselves in a permafrost area. Small lakes and rivers will freeze to the bottom and cannot be relied on, and water from easily contaminated. Large lakes and rivers, and wells drilled below

permafrost or into taliks* in permafrost, will provide a good permanent supply. Fortunately, the area we are speaking of is fairly well covered with large lakes and rivers.

All heated buildings in the North must, of course, be well insulated for protection from the extremes of temperature. There is, however, another important reason for complete insulation — that is, to prevent the passage of heat from the buildings into the ground. Any change in the thermal regime will tend to lower the permafrost line and spoil the foundation. In many cases, a false floor is erected under the building and the space is filled with moss or other suitable insulating material.

Foundations for drilling rigs may be very costly in the North. An example of this is the deep test in Alaska, in which the foundation alone cost over half a million dollars. When a deep well is to be drilled, the foundation must be such that heat from the borehole will be dissipated so that the frost level under the drilling rig is maintained or even raised. In drilling shallow wells it may be possible to move in and complete the hole before too much thawing has taken place. Where this is done, it is essential that everything possible be done to maintain the moss and vegetation covering in order to insulate the permafrost. In many cases, despite the rigours of the climate, it is easier to work in the North in winter than in summer because of the advantages of being on stable, frozen ground. New methods of transportation, such as the helicopter, coupled with improvements in light exploration equipment have partially overcome some of the difficulties of summer work in these areas. However, any large-scale oil development programme will require a great deal of time and money to overcome transportation difficulties.

No outline of the problems of permafrost would be complete without mention of the many groups now actively engaged in study and research on all aspects of development in the North. The foremost of these groups are the National Research Council, the Arctic Institute of North America, the Department of Naval Research, Washington, and the U.S. Army; all have made great contributions to our knowledge of the North. With organizations of this high calibre engaged in research and compilation of information gained from good engineering in the field, the background is provided for sound economic expansion. This expansion will bring new opportunities and new wealth to every Canadian.

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*_Taliks_: unfrozen layers in the permafrost zone.

MINING SOCIETY OF NOVA SCOTIA
DONALD J. MACNEIL, *President* S. C. MIFFLEN, *Secretary*

Dewatering an Abandoned Colliery

By LOUIS FROST *

(*Annual Meeting, Mining Society of Nova Scotia, Sydney, N.S., July,* 1952)
(*Transactions, Volume LV,* 1952, pp. 184-191)

Dominion No. 17 Colliery, of the Dominion Coal Company, Limited, is located in the district of New Victoria, N.S., at the entrance to Sydney harbour.

This Colliery was opened in 1882 to work a submarine area of the Harbour seam (see Figure 1). It was closed in 1921 and the mine, which has a make of water of approximately 250 gallons per minute, was allowed to fill.

No. 18 Colliery, situated almost a mile to the east, was opened in 1938 to work the Phalen seam in this area with the intention, later, of driving a cross-measure tunnel to mine the section of the Harbour seam lying between No. 17 Colliery and the Princess Colliery of the Nova Scotia Steel and Coal Company, Limited.

*Chief Mining Engineer, Dominion Steel and Coal Corporation, Limited, Sydney, N.S.

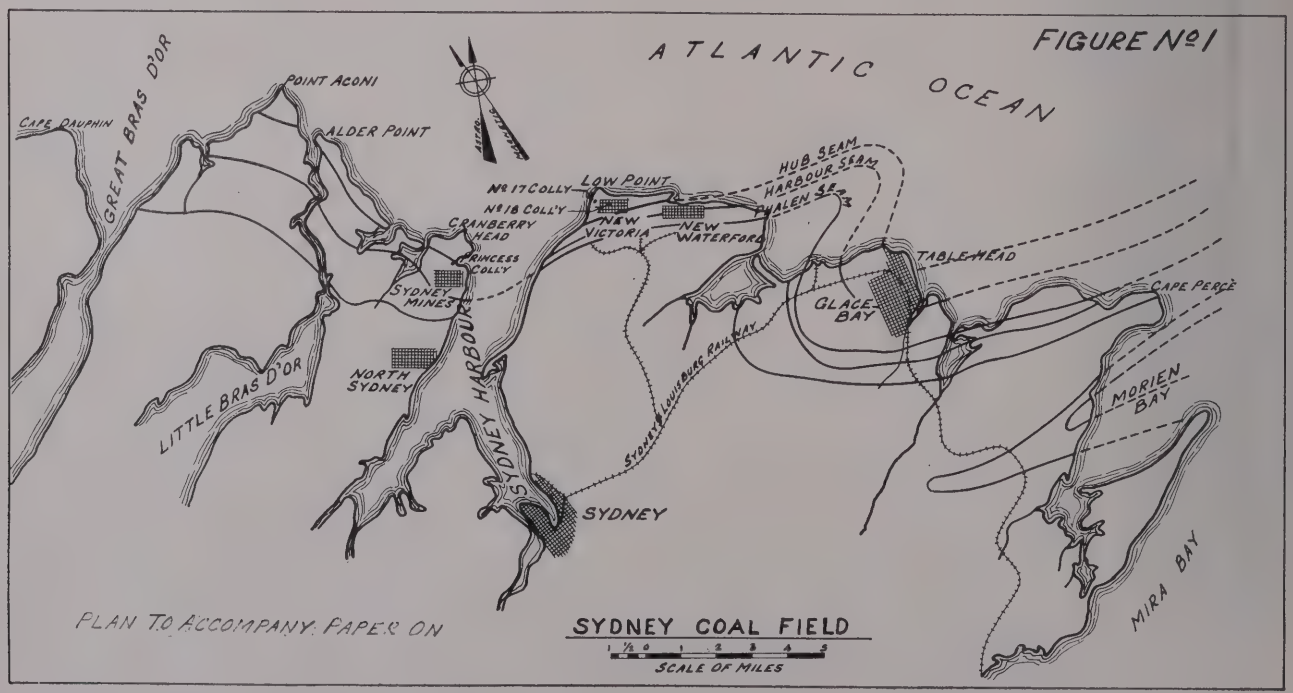
In 1945 it became necessary to open this area in the Harbour seam and a cross-measure tunnel was driven from the existing workings of No. 18 Colliery in the Phalen seam, which underlies the Harbour seam at a depth of 500 feet.

The tunnel was started in 1946 and completed in 1948. The cost of opening this section, which contained only 6½ million tons of coal, made imperative the recovery of all the available coal and precluded the maintenance of the large barriers that had been left in No. 17 Colliery for impounding the water, especially as the maintenance of these barriers would have meant the loss of 885,000 tons of coal, or 12.3 per cent of the whole tonnage available. Therefore, a study was made to determine the best means to dewater the 250 million gallons of water lying in the old No. 17 Colliery.

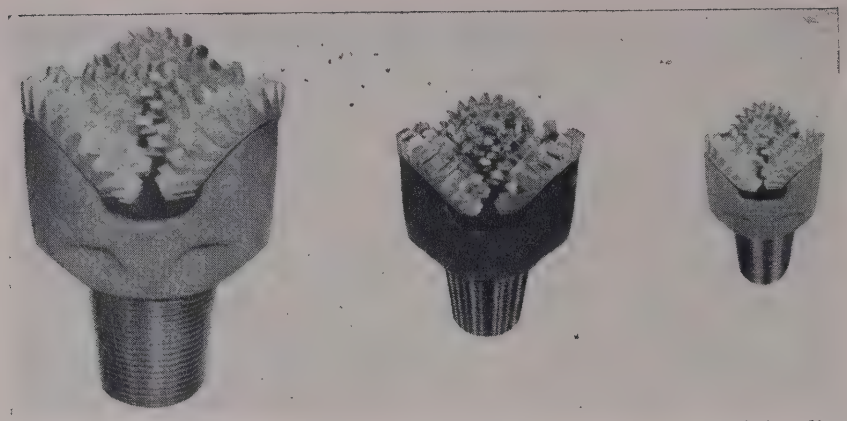
It soon became apparent that the dewatering of No. 17 Colliery could not, at a reasonable cost, be undertaken from the surface, with sinking pumps following the water as it receded, because, as far as could be ascertained, the roof supports in the main entries in No. 17 Colliery had been withdrawn and the roof allowed to fall. Therefore, because of the very high cost it would have involved, this method of approach to the problem had to be discarded.

As the Company had, on a previous occasion at Springhill (1) successfully dewatered a flooded area overlying active workings at a vertical distance of 400 feet with a water pressure of 530 pounds, it was felt that No. 17 Colliery could be dewatered in this manner, despite

(1) *Boring to Upper Seam, No. 2 Mine, Springhill,* by T. L. MCCALL; *Trans.,* C.I.M. and M.S.N.S, Vol. XXXII, 1929, pp. 458-468.



DOM. Nº 18 COLLIERY
FIG. Nº 2
SCALE IN FEET
OUTCROP PHALEN SEAM
OUTCROP HARBOUR SEAM
DOM. Nº 18 DEEPS
BOREHOLE Nº1 VERTICAL
BOREHOLE Nº2 VERTICAL
DOMINION DEEPS
BOREHOLE Nº3 HORIZONTAL
PRINCESS WORKINGS



7¾ in. dia. 5¾ in. dia. 3¾ in. dia.

Figure 3.—Tricone bits.

the fact that vertical boring for so great a distance, 500 feet, had not been attempted here in the past.

It was realized that to reach a point where the inundated workings could be tapped successfully would entail very great care to prevent drift of the borehole. Consequently, as the workings in No. 17 Colliery were on a pillar-and-room system with only limited sections of pillars drawn, it was decided to aim at a small section of the workings where the pillars had been drawn (see Figure 2).

In the vertical drilling at Springhill, which was under the supervision of experts imported from Britain for the purpose, a diamond drill of specially robust construction had been used. This entailed the withdrawal of the cores at 9-foot intervals and considerably slowed down the job.

To speed-up the No. 17 operation, it was decided to use continuous drilling with Tricone bits (see Figure 3), an innovation in coal-mine drilling, although bits of this type had been used previously in drilling oil wells. Doubt was expressed in many quarters as to the possibility of maintaining a proper vertical or horizontal course with this type of bit. The results obtained, however, justified all expectations.

The total static head at the foot of No. 17 Colliery Deeps was 1,117 feet. It was therefore decided to dewater in three stages, so that at no time would the pressure at the point of break-through into the old workings exceed 410 pounds. To this end, three points were selected (see Figure 2). The first hole was started on April 21st, at No. 3 level, where the calculated static pressure would be 300 lb.

In the previous dewatering at Springhill in 1929, only a 20-foot case-pipe had been used at the bottom of the borehole, with the result that the dewatering operation scoured the hole so severely that it impeded the dewatering and finally the hole had to be abandoned.

To avoid a similar happening in this case, where the strata consisted mostly of soft shales, it was decided to case the hole throughout in two operations (see log of strata, Figure 4).

Bits of three sizes were used — 7¾-, 5¾-, 3¾-inch. The plan of the operation was to bore the first section past a small seam of coal which lies 110 feet above the Phalen seam, with a 7¾-inch bit, and to cement-grout a 6-inch case pipe to this height. The pipe, although cement-grouted into place, was also supported on 8 in. by 10 in. pine beams, securely hitched into the ribs, and, to ensure complete consolidation of the strata in the vicinity of the borehole, fifteen feet of the roof on each side of the borehole was cement-grouted to a height of 12 feet. Figure 5 shows method of drilling.

When the first pipe was grouted into place, a test pressure of 1,000 pounds per square inch was applied on the pipe and the fittings to ensure absolute security in the event of a cross-break in the strata.

In drilling the first section, the bit was followed by a 5-foot core barrel, and dog nuts of nearly the same diameter as the bit were set at every third rod to provide rigidity to the rods.

A careful survey was made during the advance of the hole to determine drift, if any. This was done by inserting a glass cylinder filled with gelatine into the hole at frequent intervals. The gelatine was allowed to set and the angle of the set thus indicated drift from the original line. It was found, how-

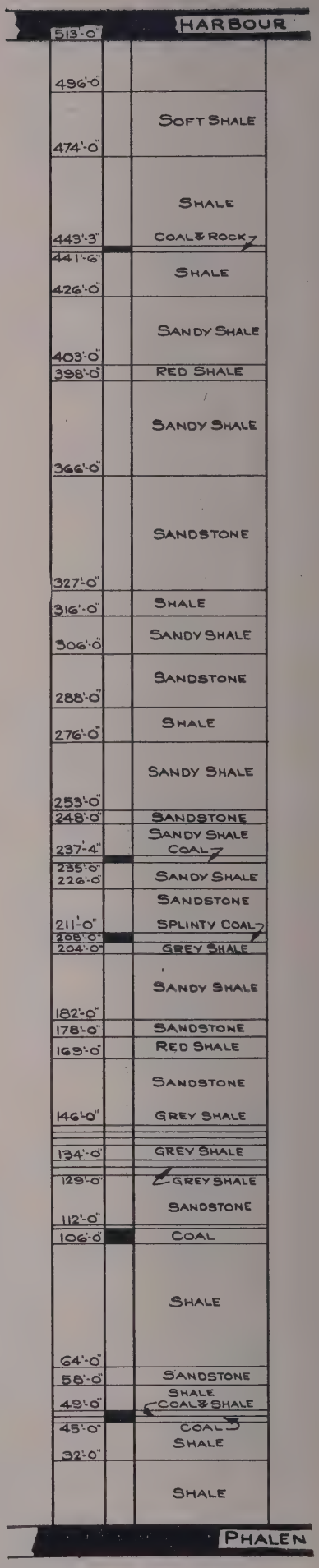


Figure 4.—Log of strata.

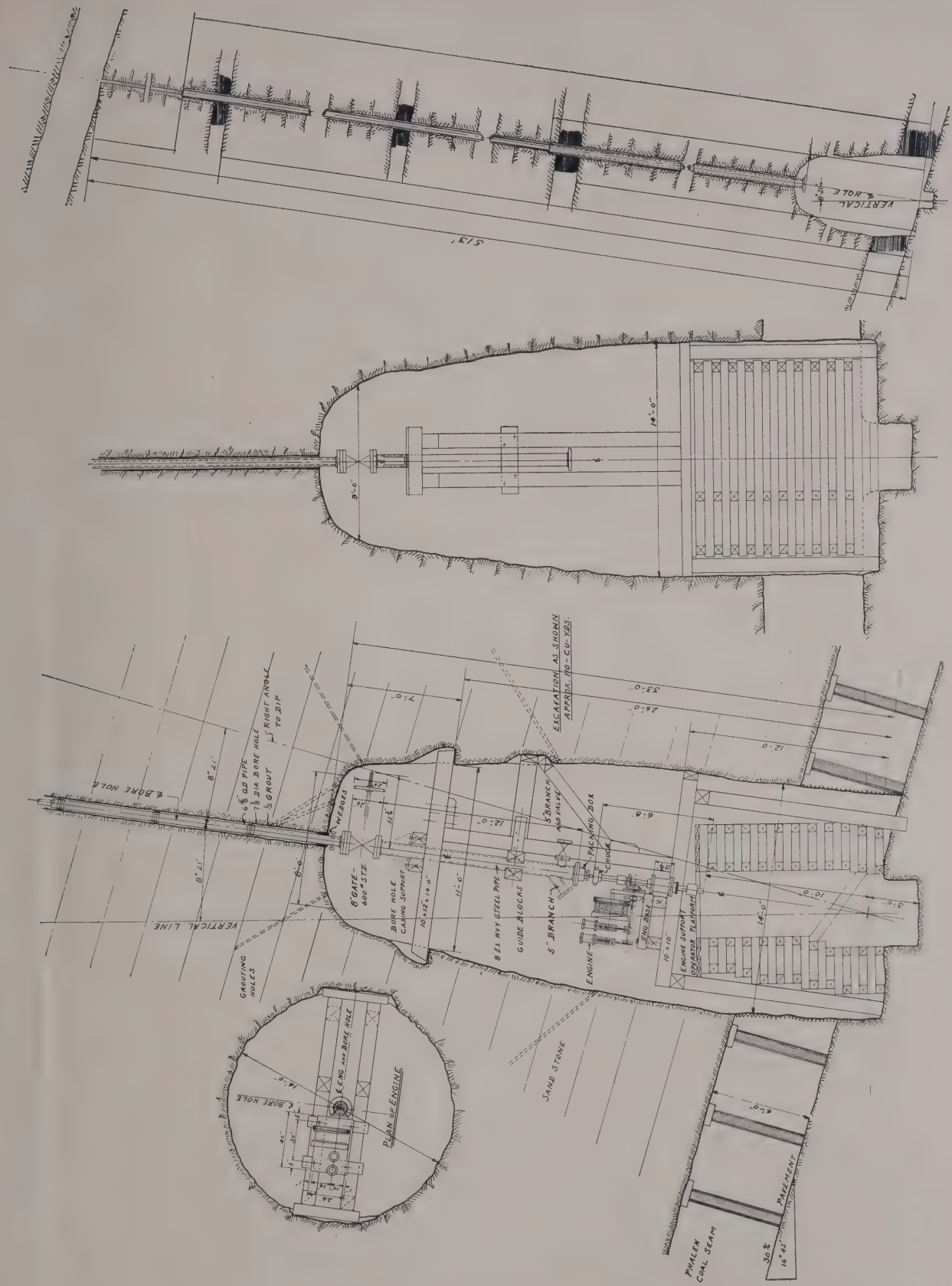


Figure 5.—Arrangement of diamond drill for dewatering No. 17 mine (Harbour seam). Borehole up from No. 18 mine (Phalen seam).

ever, that, in drilling the first section, the hole had not deviated at all from the original sight line.

The second section of the hole was bored to a height of 450 feet, with a 5¾-inch bit, and again, to prevent whipping of the drill rods, dog nuts 5¾ inches in diameter (see Figure 6) were spaced at 50-foot intervals behind the bit and, to ensure that the bit would not drift off course, a short core barrel, arranged with suitable openings to permit the débris to flow past, was placed directly behind it.

When the drilling of the hole to the above mentioned height was completed, a 4-inch pipe was cemented into place, and the remainder of the hole was bored with a 3¾-inch bit. The total distance bored to the break-through was 513 feet.

To ensure that the drill rods would not be driven back by pressure when the water was encountered, a special yoke was fitted on

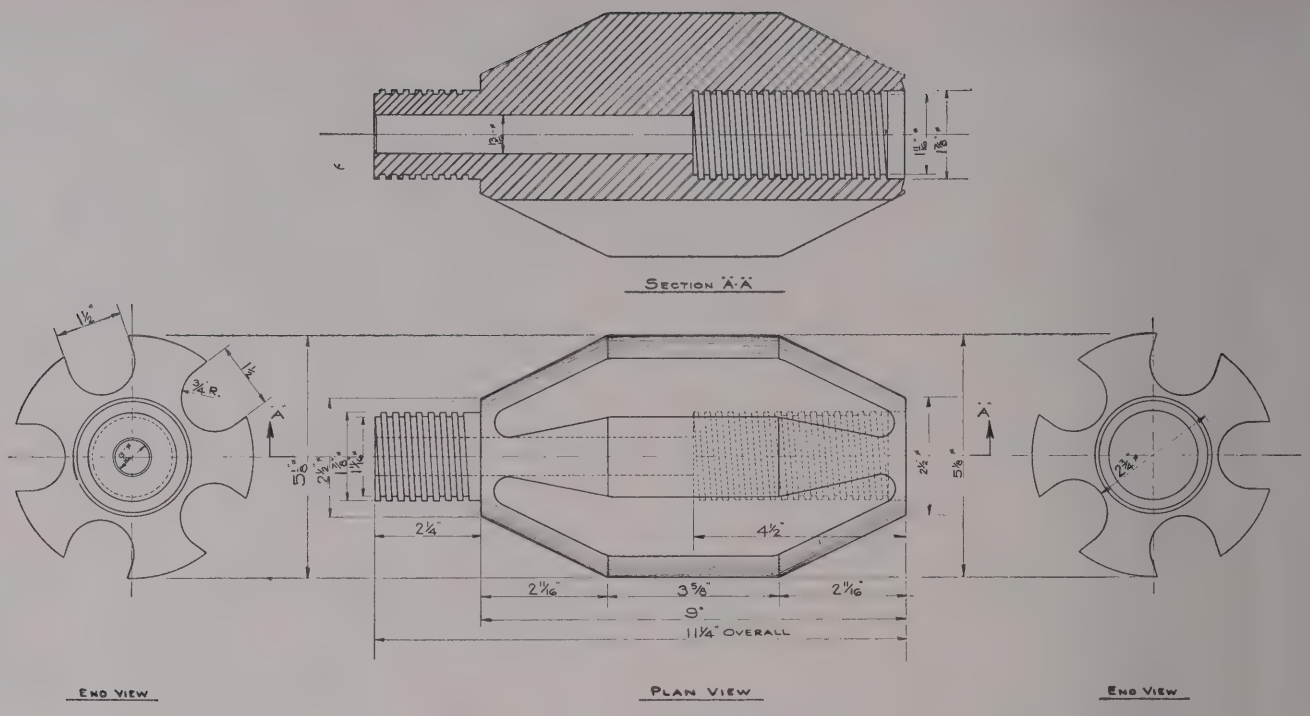


Figure 6.—Plan and section of dog-nut.

the end of the drill rods and held taut with ropes from an air winch, to take the shock. This was an important precaution, as prior experience at Springhill had shown that the pressure of the water could drive the drill rods out of the hole with great force.

The best drilling footage obtained in this vertical hole was 15 feet in one shift of eight hours; the average advance was 12½ feet per shift. This was a vast improvement over progress made here previously in diamond-drill holes, which had averaged only 10 feet on a two-shift operation. Even at that, the drilling speed could have been increased, but, to ensure absolute accuracy, it was held to a point considered adequate. In another hole, drilling vertically downward, 42 feet 6 inches was bored in an eight-hour shift.

The drilling of this hole was a complete success and a survey made at the completion of 450 feet of drilling showed a drift of only ½ of one degree.

The cost of boring and casing this hole is shown in Table I.

TABLE I.—COST OF BORING AND CASING VERTICAL HOLE
(Per foot)

	LABOUR	MATERIAL	TOTAL
Boring....	$ 9.80	$ 1.75	$11.55
Casing borehole.	3.66	2.00	5.66
TOTALS.	$13.46	$3.75	$17.21

From the initial break-through, the flow of water was 1,050 gallons per minute although the rate dropped on two occasions to 600 g.p.m. This was soon rectified when the hole was flushed back by by-passing the discharge column directly into the borehole.

This hole dewatered a total of 98,976,000 gallons in a period of 97 days. The total amount of water pumped from this borehole amounted to 154.5 million gallons.

In the meantime, boring of a second hole was under way at No. 4 level, where the static head was calculated to be 410 lb. per square inch (see Figure 2).

In setting off this borehole, a section was chosen where the old mine plans indicated that the pillars had been drawn. As the interval between the seams was already known from the drilling of the borehole at No. 3 level, no exceptional difficulties were anticipated.

In boring this hole, the same procedure was followed as in the first hole and boring proceeded successfully to the same height, 513 feet; but as, at this height, the hole had not penetrated the seam, a survey was made to determine if the hole had drifted. As in the case of the first hole, however, this survey showed a drift of less than one degree or a horizontal drift of eight feet, which brought it well within the area selected.

Drilling was continued and the Harbour seam was penetrated at a depth of 567 feet. This indicated a thickening of the strata in this area and that the hole had entered a solid pillar of coal rather than the waste.

As the old mine plans indicated a pillar size of 60 ft. by 30 ft. in this area, it was decided to try and burst the pillar with a heavy charge of explosive and, after consultation with Canadian Industries, Limited, through their Maritime representative, it was agreed to use commercial nitro-glycerine, 100 per cent strength, in solid form, similar to that used in blowing in oil wells.

The problem, however, was to devise a method of placing and holding the explosive in place during the firing at this vertical distance (567 feet) above the seam from which the boring had been carried on and, although an expert in oil-well blasting was supplied by the courtesy of Canadian Industries, Limited, he had never had experience in up-blasting and could not advise on the procedure to follow.

However, the operation was successfully accomplished by a method devised by ourselves (see Figure 7) and one which is undoubtedly an innovation in the field of borehole blasting. It is described below.

A canister of five-gauge plate, 12 feet long and 3 inches in diameter, was prepared in the form of a torpedo to hold the nitro-glycerine and tamping material. The canister was in two parts; one part 8 ft. 9 in. long, 8 feet of which was to carry the explosive, and the other 4 feet long to carry 2 feet of clay and 2 feet of a specially constructed plug of hardwood.

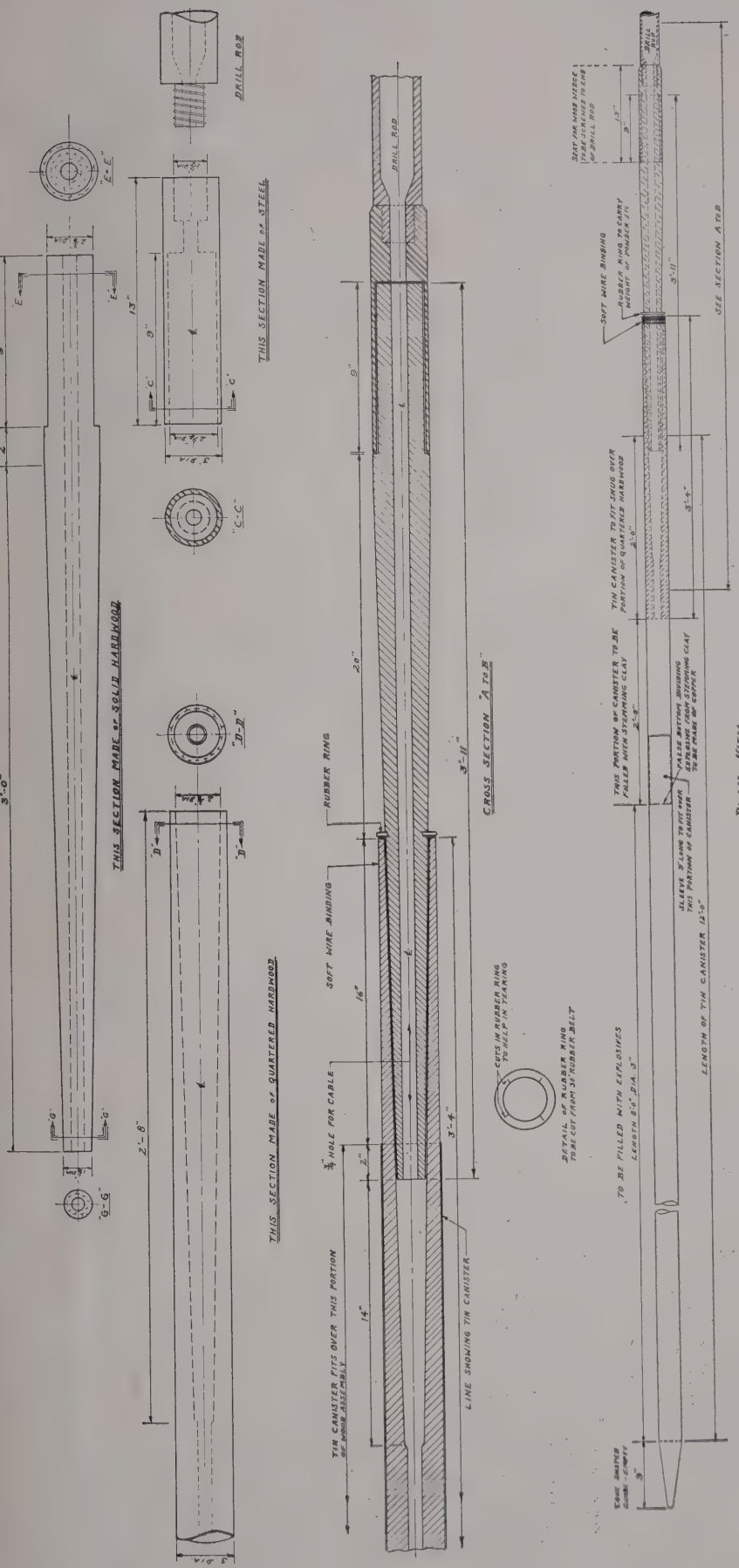

THIS SECTION MADE OF SOLID HARDWOOD
THIS SECTION MADE OF STEEL
THIS SECTION MADE OF QUARTERED HARDWOOD
DRILL ROD
RUBBER RING
SOFT WIRE BINDING
HOLE FOR CABLE
TIN CANISTER FITS OVER THIS PORTION OF WOOD ASSEMBLY
LINE SHOWING TIN CANISTER
CROSS SECTION "A T-B"
DETAIL OF RUBBER RING
CUTS IN RUBBER RING TO HELP IN TEARING
TO BE FILLED WITH EXPLOSIVES
LENGTH OF TIN CANISTER
PLAN VIEW
SEE SECTION A T-B

Figure 7.—Sketch showing plant detail of canister and wood wedge for placing charge in borehole drilled up in strata for dewatering Harbour seam.

This plug was split lengthwise into four parts and wound with fine copper wire sufficient to hold it together, but which would break at a pressure of 10 lb. The plug was hollowed out in the centre to take a cone-shaped wedge. The wedge was driven into the plug and as soon as the pressure it exerted on the latter exceeded 10 lb., the copper wire snapped and the sectional pieces of the plug were forced tightly against the wall of the borehole, securely anchoring the charge in place. On test, it was found that the cone-shaped nose of the torpedo collapsed with a pressure of 20 lb. This ensured the anchoring of the torpedo securely before any pressure was applied to the explosive contained in the cylinder.

The firing cable was threaded through the retaining plug and the drill rods. Also, to make certain that the detonators would not pull out and that the cables would not break, the cable was suspended on a nylon cord which was fastened to the inside of the canister.

Thirteen-and-a-half pounds of 100 per cent glycerine made up the charge and, to ensure firing the explosive, six detonators were used. These were imbedded in three plugs of forcite, 40 per cent strength.

To avoid any accident in the event of a blow-back, the borehole casepipe was extended back into a crosscut between the main and the low level, and the end of the pipe was heavily covered with bags of stone dust.

The charge was successfully exploded with a 50-shot battery. This type was used because of the firing distance from the battery. The only evidence of a successful explosion was the very heavy thump in the roof, the appearance of portions of the plug and firing cable at the bottom of the borehole, and the effect of the back pressure of the blast in spreading the stone dust in the crosscut.

An indication of the force of the explosive blast of 13½ lb. of nitroglycerine was obtained when one pound of nitro-glycerine was exploded on the surface. This charge was fired at a distance of 100 feet from the winding engine house at the colliery. The blast shook this building so violently that the operator thought it was going to collapse.

The attempt at bursting the pillar failed. The assumpion is that the thickening of the strata is local, indicating a sharp disturbance in the Harbour seam, and that in this section of the mine, although shown as

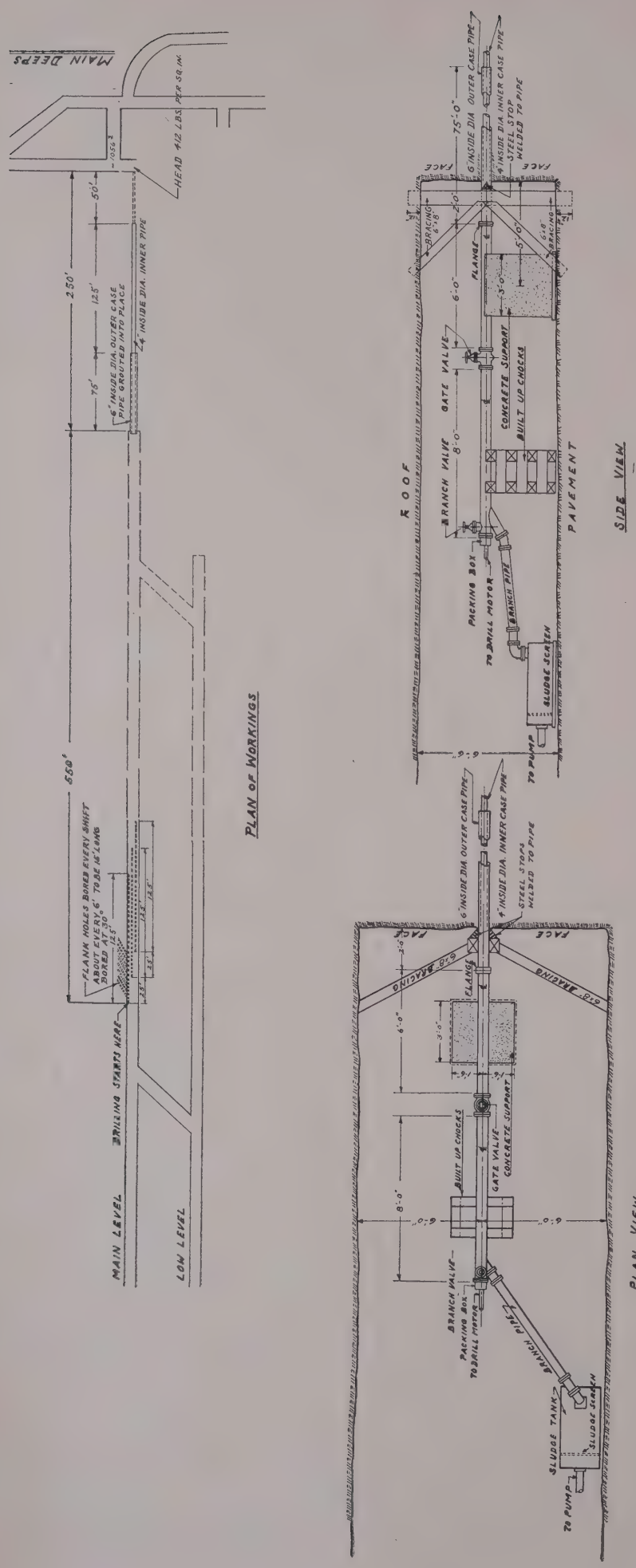


Figure 8.—Sketch showingproposed method of de-watering No. 17 colliery from main level.

drawn, the pillars may not have been worked at all.

Under these circumstances, it was decided to dewater the rest of the workings of No. 17 Colliery from the workings in the Harbour seam opened by the cross-measure tunnel from the Phalen seam, which had progressed to a point where it was possible to bore from the main levels in the new Harbour seam workings, to tap the old main deeps in No. 17 Colliery (see Figure 2).

Horizontal Drilling

The Harbour seam in this section is 6 feet thick, and the main levels were driven to about 250 feet from old No. 17 Colliery. A calculation, made by the writer, on the probable strength of the barrier, indicated that this thickness of pillar gave a factor of safety against the static head of water of 6.7 to 1. This was considered well within safe working limits. However, to provide for any unforeseen contingency, the face of the pillar and of the surrounding strata was consolidated by cement-grouting the coal, the roof, and the floor to a depth of 13 feet.

The drill was set up on a sight line to give a vertical rise of 2 degrees and a horizontal angle of 3 degrees so as to strike the deeps at a point as near the centre of the seam as possible. To ensure extra care, a survey of the borehole was made every 10 feet by placing a tube filled with gelatine in the hole and allowing it to set.

The method of boring the horizontal hole is shown in Figure 8.

It was planned to drill 60 feet and to grout a 6-inch case pipe into place to this depth.

Drilling was commenced with a 7¾-inch Tricone bit and, to avoid the weight of the bit causing drift toward the floor of the seam, a 5-foot follower 7½ inches in diameter was placed behind the bit. This was followed with dog nuts, 7½ inches in diameter, at intervals of 9 feet along the length of the rods.

After this hole had been drilled to a distance of 60 feet, a careful survey indicated a total fall of 3 degrees in the last 15 feet of the hole.

To bring the bit back to proper elevation, a cement grout, consisting of 1 part cement and 1½ parts sand, was pumped into the hole and allowed to set and, in order to avoid a recurrence of this drift toward the floor, it was decided to give the drill a vertical inclination of 3 degrees,

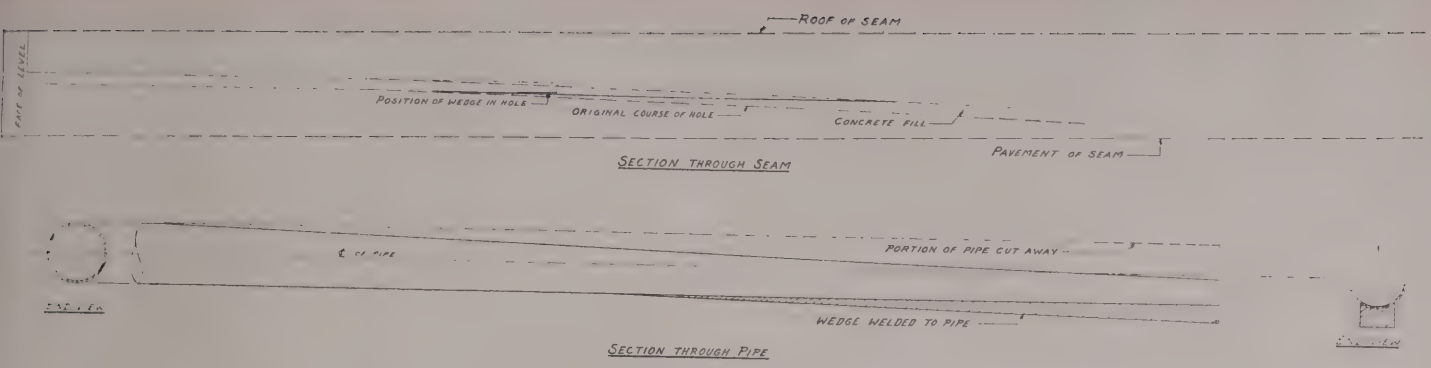


Figure 9.—Sketch showing method used to maintain grade of horizontal hole.

To accomplish this, the last fifteen feet of the pipe was cut off at the top on this angle, and a wedge, shaped to the pipe to prevent it from turning, was welded on the bottom of the cut portion of the pipe (see Figure 9).

This case pipe was placed and grouted into position. A leather plug on the drill rod prevented the grout from running out. Additional grouting holes were so placed that the grout would flow between the pipe and the coal side of the hole.

The grouting plug was withdrawn before the cement had fully hardened, and the case pipe carefully bored out and cleaned of cement. When the cement had fully set, hydraulic test pressure of 1,000 lb. per square inch was put into the borehole and all fittings.

Boring was continued with a 5¾-inch bit to a depth of 200 feet and in this case also a short barrel, 5½ inches in diameter, was used to prevent the bit from dipping. Dog nuts, 5½ inches in diameter, were also spaced at intervals of 27 feet to prevent the rods from sagging.

Frequent gelatine surveys indicated that the borehole was being maintained on correct alignment and, during the drilling for the 4-inch-diameter case pipe, no exceptional difficulty was met.

When the hole reached the depth indicated above, a 4-inch case pipe was grouted into place in the manner already indicated.

Here, also, all the precautions already described were employed to prevent the drill rods being forced out of the hole by the pressure of the water, which, on break-through, was found to be 420 lb. per square inch.

This hole has drawn off the water at the rate of 950 gallons per minute. A few days after the hole broke through, the flow dropped to 700 gallons but, after the column was flushed back, the hole continued to discharge the full amount of 950 g.p.m.

When the static pressure dropped to 90 lb., a test was made to ascertain if the workings had been fully dewatered. The pump was allowed to stand for several days and the static pressure noted and, as the 'make' in No. 17 was known, the approximate position of the water in the deeps could be determined. The pump was then re-started and the water lowered to the previous point.

This colliery has now been dewatered to the point where the operation can be carried on in this area. The total amount of water pumped to date amounts to 250 million gallons and the footage bored to dewater the colliery totals 1,395 feet. The cost of $24,020 is a low figure which could not have been approached in an attempt to dewater the mine by cleaning the main entries in No. 17 Colliery.

In conclusion, I wish to acknowledge gratefully the assistance received from the staff of the Mining Engineering Department of the Dominion Coal Company in the preparation of the drawings accompanying this paper.

MINING SOCIETY OF NOVA SCOTIA
DONALD J. MACNEIL, *President* S. C. MIFFLEN, *Secretary*

Conjectural Shape and Extent of the Sydney Coalfield

By T. BINNERT HAITES

(*Annual Meeting, Mining Society of Nova Scotia, Sydney, N.S., July,* 1952)
(*Transactions, Volume LV,* 1952, pp. 192-202)

INTRODUCTION

SHAPE and extent of the Sydney coalfield are in first instance determined by the dimensions of the former basin of deposition. This three-dimensional body became distorted as a result of post-depositional structural changes. Last but not least, erosion was responsible for the finishing touches.

The results of these three stages in the history of the field — deposition, tectonics, and erosion — are superimposed on each other and therefore, nowadays, sometimes difficult to unravel. For instance, splits in the seams which are truly depositional phenomena have in the past sometimes been mistaken for complicated faults. Structural features are usually common to several seams in a succession, whereas every seam has its own depositional characteristics. Erroneous conclusions may also be drawn from casual observations of the topography. Parts of the Sydney coalfield exhibit an inversion of relief, with the structural highs, *e.g.*, the Bridgeport anticline, occupying an indentation of the shore line, and the structural lows, *e.g.*, the Glace Bay syncline, occurring on a promontory.

Studies of the same nature as the present one were skilfully advanced by F. W. Gray in 1933 (8) and by F. W. and R. H. Gray in 1941 (9). After ten years, the knowledge of the Sydney coalfield has increased considerably and geological science has developed some new methods of research which have proved helpful in the interpretation of the phenomena on which the present conclusions are based. Although the 'probable' and 'possible' reserves have changed and will continue to do so with the increase in knowledge, mineability requirements have largely remained the same as outlined by Gray sr. and jr., reference to whose paper will give further detail.

PALAEOGEOGRAPHICAL ANALYSIS

Plant fossil evidence underlies Bell's correlation of the Upper Westphalian strata in the Inverness and Mabou coalfields (Inverness group) with those of the Sydney coalfield (Morien group) (1, Fig. 11). Carrying the analogy a step farther, the observation is made that a striking resemblance exists between the Seven-foot seam in Mabou Mines, the Seven-foot seam in Inverness, and the Harbour seam in the Sydney coalfield (see Table I). The Harbour seam was found to be characterized by two bone horizons of which the upper one is well developed in the Sydney Mines district (12, p. 8 and encl. H. 65). The coal separating the two bone horizons averages two feet in thickness, whereas in the Seven-foot seam in Inverness No. 3 mine this figure is 1 ft. 5 in.; in Mabou, 1 ft. 6 in. was measured by Gilpin (6, p. 14) in the equivalent interval. It proves that, in these areas, swamp development was interrupted at approximately the same time intervals.

The whole stratigraphic column of the Sydney field compares well with that of the Inverness field, but not as well with that of the Mabou field (see Figure 2). The major discrepancy is the presence of the Eight-foot seam in Mabou Mines or the absence of a similar thick coal horizon in Inverness and Sydney. Possibly, the Eight-foot seam passes laterally into shale.

The suggested correlation does not appear to be in accordance with Bell's zoning of the Inverness strata in Mabou Mines and Inverness (2, p. 31). However, Bell, in an oral communication to the writer, stated that the separating line between the

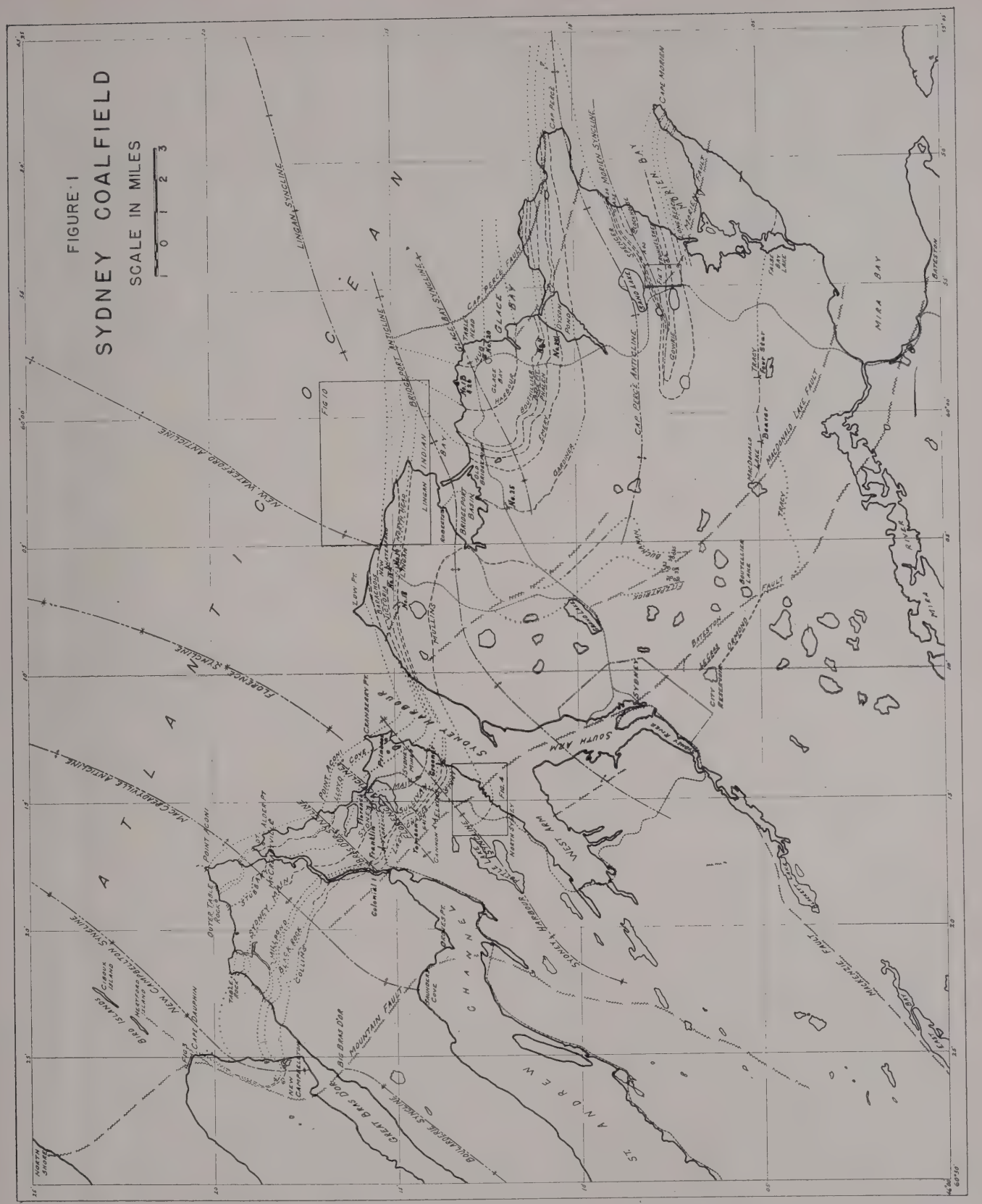
TABLE I.— CORRELATION OF SEAMS AT MABOU, INVERNESS, AND SYDNEY

MABOU MINES	INVERNESS	SYDNEY
7-Foot seam*	7-Foot seam	Harbour seam
1 ft. 6 in. coal 1 in. shale	1 ft. 8 in. coal 1½ in. bone	2 ft. 0 in. coal 2 in. bone & coal
1 ft. 6 in. coal 1 in. shale	1 ft. 5 in. coal 1 in. clay	2 ft. 0 in. coal ½-3 in. bone, shale, & coal.
2 ft. 0 in. coal 1 in. clay	3 ft. 0 in. coal 1 in. bone	2 ft. 0 in. coal 1 in. bone
1 ft. 9 in. coal shale		1 ft. 0 in. coal 1 in. bone
8-Foot seam (?)	10 in. coal	6 in. coal

*This section differs from the one obtained during the summer of 1951 in the old slope. Presumably, Gilpin's section was taken in the underground workings, and as such it may represent a better average of the seam.

*Geologist for Coal Operations, Dominion Steel and Coal Corporation, Limited, Sydney, N.S.

(8) For references, see end of paper.



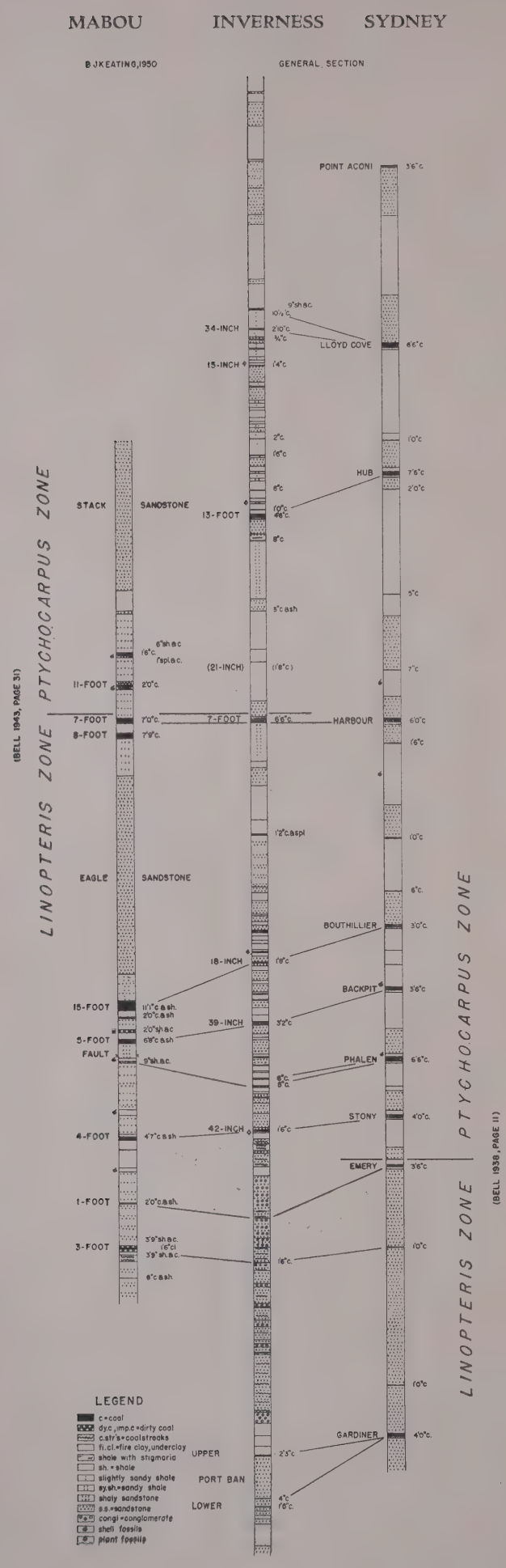
Ptychocarpus unitus and the *Linopteris obliqua* zones is rather arbitrary and could be lower in the sections of those localities so as to coincide with its position in the Sydney field.

The detailed correlation of the Seven-foot and Harbour seams, as well as the tentative correlation of the stratigraphic columns, leads to the following conclusions:

(1) Since, in Inverness, the Broad Cove formation overlies the coal measures with a probable conformity (17, p. 49), the Point Aconi seam in the Sydney field may represent the highest seam in the section.

(2) The Inverness and Morien strata were likely deposited in one basin.

This basin was not of exceptionally large size. In western Europe, the double seam Grauweck-Senteweck was found to occur in the Belgian (Kempen), the Dutch (South Limburg), and the west German (Aachen) coalfields, which involves a lateral extension of 50 miles (10, 22). The Katharina index coal bed in the Rhenish-Westphalian coal basin has a proved length of 140 miles and a width of 35 miles in Westphalia (23, p. 191).



In the Yorkshire-Derbyshire-Nottinghamshire coalfields, the Warran House-Barnsley-Top Hard seam has an areal extent of 60 by 80 miles (H. S. Haslam, oral communication).

The question may be asked whether an overland connection existed between the Inverness and Sydney fields or whether these areas were parts of a string of basins flanking an upland. It has been argued by other students that the present boundaries of the Sydney coalfield are structural as well as sedimentary. F. W. Gray (oral communication) noted the occurrence of biotite in arkosic sandstones of the fossil Harbour river on the northwest side of Florence colliery (13, Figure 2) and interpreted this as an indication for a nearby source rock (see also 9, p. 295). Bell can be quoted similarly: 'The Morien series was deposited in a subsidiary basin of deposition that was bordered on the northwest by a 'positive' area upon which it gradually encroached. This old 'positive' area is represented today by the northern tableland of crystalline rocks" (1, p. 9).

However, the Morien strata nowhere exhibit a lithology comparable to the Upper Windsor fan conglomerates in the area under consideration. At Cape Dauphin, they can be seen overlapping crystalline rock and are found to be composed of brecciated granite-material (see Figure 3). The younger limestone and gypsum deposits suggest that most of Cape Breton became submerged under the shallow Windsor sea and that, therefore, the relief at the start of the Pennsylvanian was low. Again, the Canso deposits at Cape Dauphin do not indicate that their components originated from the nearby uplands. Except possibly for a few monadnocks* (21), these uplands seem to have been absent during Pennsylvanian times. Consequently, the conclusion drawn by F. W. and R. H. Gray (9, p. 297) that "much sediment was washed into the subsiding basin of deposition by streams flowing easterly from the northern tableland of Cape Breton" does not seem to be substantiated by the composition of the majority of the Morien strata.

The coal seams of the Inverness or Morien groups do not exhibit features suggesting that, within the limits of the present fields, the edges of the former swamps were reached. There are splits and wash-outs in these seams, indicative of the presence of fossil rivers. The size of these rivers and their floodplains suggests that their headlands could not have been nearby. Similar evidence is afforded by the coal distribution in the seams. In the Sydney coalfield the Backpit seam† can be traced from New Campbellton to Port Morien with a fairly uniform thickness of three to four feet. Little is known of the Six-foot seam at New Campbellton, but in the brook where the seam was first discovered it consisted of 3 ft. 6 in. coal, 6 in. fireclay, and 2 ft. 0 in. coal (N.S. Dept. of Mines, Ann. Rept., 1894, p. 27.) This proves that the Phalen seam does not peter

*Such a monadnock may have existed in the Inverness district, where "strata equivalent in age to a middle part of the Morien series show partial derivation from rocks of the Mabou Highlands" (1, p. 9).

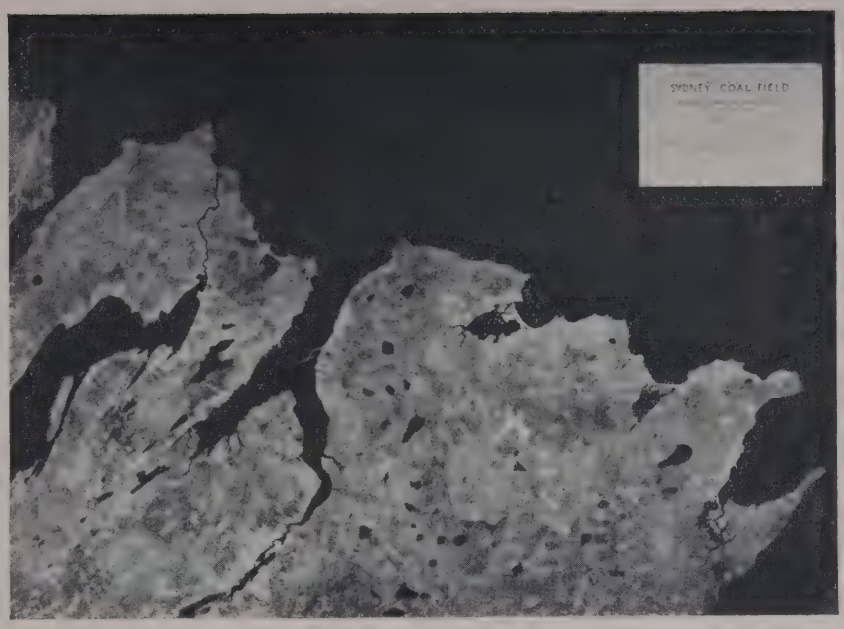
†The correlation table published earlier (13, p. 217) has been corrected for the New Campbellton district (misprinted as New Carleton district). Stratigraphical and coal-petrographical evidence suggested the Four-foot seam to be the equivalent of the Backpit seam; consequently the Six-foot seam is now thought to be the equivalent of the Phalen seam.

←—**Figure 2.—Sections of the coalfields at Mabou, Inverness, and Sydney.**

SCALE IN FEET
50 0 50 100 150

out in this district. The absence of the Mullins and Tracy seams, the equivalents of which have not yet been traced on Boularderie island, could be attributed to a structural rather than a depositional reason.

At North Sydney, the Mullins seam is washed out by a fossil river, north and west of which the seam has not yet been found (see Figure 4.) In the New Waterford district, this seam is 6 feet thick. However, it has not been traced with certainty south of the Bridgeport basin. On the other hand, the Gardiner seam is known only from the Glace Bay district and has not been discovered in the New Waterford district, except for Robert's pit on the north shore of the Bridgeport basin. In this area, both the Mullins and Gardiner seams have a 'rider' seam and their thicknesses and stratigraphic positions match roughly. Moreover, their sections show certain coal-petrographical similarities (P. A. Hacquebard, oral communication).



Aerial mosaic of Sydney coalfield.

Figure 3.

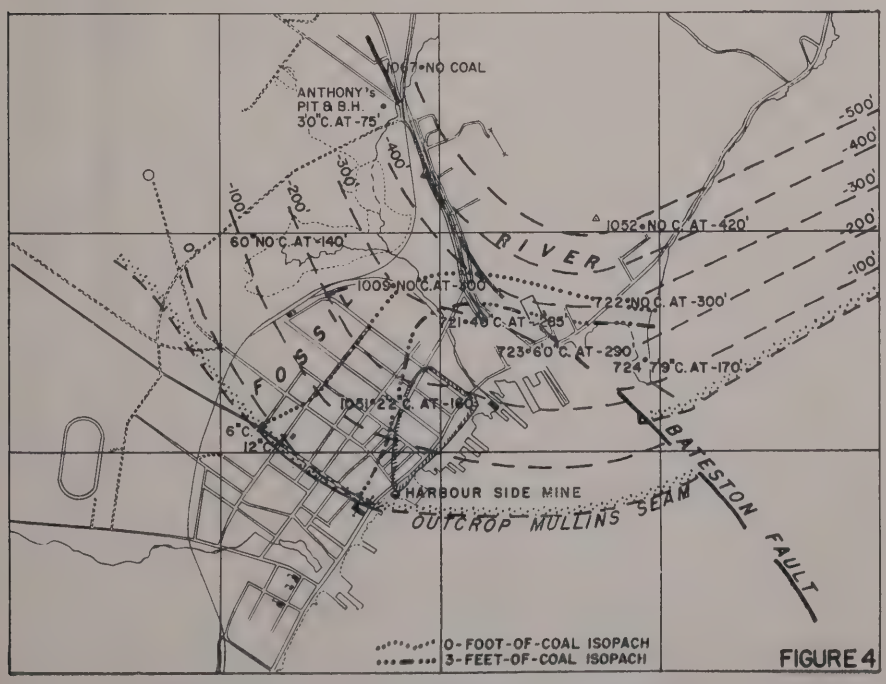


Figure 4.—Plan of the Mullins seam at North Sydney.

The Tracy seam is the lowest mineable seam that is known in the Sydney coalfield. It has been traced — although not continuously — from False Bay beach westward to Macdonald lake, where it is probably cut off by a major fault (see page 297). Over this distance it was found to have a very peculiar composition (P. A. Hacquebard, oral communication). So far, these characteristics have not been discovered in any of those coals which, according to Bell, are to be correlated with the Tracey (16, Map 361*A*). Consequently, this seam is still unknown in the area west of the 60°5′ meridian.

So far, the persistence of the coal-horizons only has been considered. Quantity and quality of the seams were not discussed. A river, running through the Pennsylvanian

swamps, can be expected to have had similar abundance of vegetation on both borders. Translated into coal terminology, this means that a coal seam encountering a wash-out should regain its height and quality on the opposite border. However, there may be deviations from this rule, one of which is embodied in the sulphur and ash distribution, discussed in a previous contribution (13, p. 220).

The problem of the symmetry or asymmetry of the border lands of these fossil rivers confronts mining operations in the Sydney coalfield at an increasing rate. Evidence for a recovery of the Harbour seam beyond the fossil river on the north-side of Florence colliery has so far been negative, except for bore-hole *D*-7 at Alder point, in which 6 ft. 6 in. coal was found. The same applies to the Mullins seam, of which not even the horizon has been traced on Boularderie island. The big Phalen wash-out in Dominion Nos. 16 and 18 collieries intersects the crop of this seam, east of Old Victoria. A partial recovery on the opposite border of this fossil river is exhibited by the section on the shore:

1 ft. 9 in. coal	
	2½ in. shale
11¼ in. coal	
	4½ in. shale
9 in. coal	
	3 in. shale
3¾ in. coal	

The only mineable Phalen coal to the Sydney Mines district is extracted in Colonial No. 1 colliery, but it is not known how this body of coal is situated with respect to the above mentioned river channel. The same applies to the Six-foot seam in New Campbellton, of which only very few records are available.

Similar remarks may be made about the Hub, Emery, and Gardiner seams. Submarine boreholes to the last mentioned seams failed to indicate mineable heights of coal.

It is encouraging to note that there is one instance of a large and extensively mined body of coal on the seaward side of a river channel. In the abandoned Bridgeport and Lingan collieries, a split occurred in the upper part of the Phalen seam, separating a three-feet-thick rider seam from the six-feet-thick main seam. This split opens to the west, suggesting a river channel situated somewhere in that direction, in a part of the seam presently removed by erosion. The coal to the east has been mined ever since the early days of the Sydney coalfield.

In 1933, F. W. Gray (8, p. 5) advanced the idea of progressive upward enlargement of the mineable areas in the seams. This theory was further worked out in 1941 by F. W. and R. H. Gray, who assumed the original coal-bearing horizons to be lens-shaped, with the convex side downward (9, p. 317).

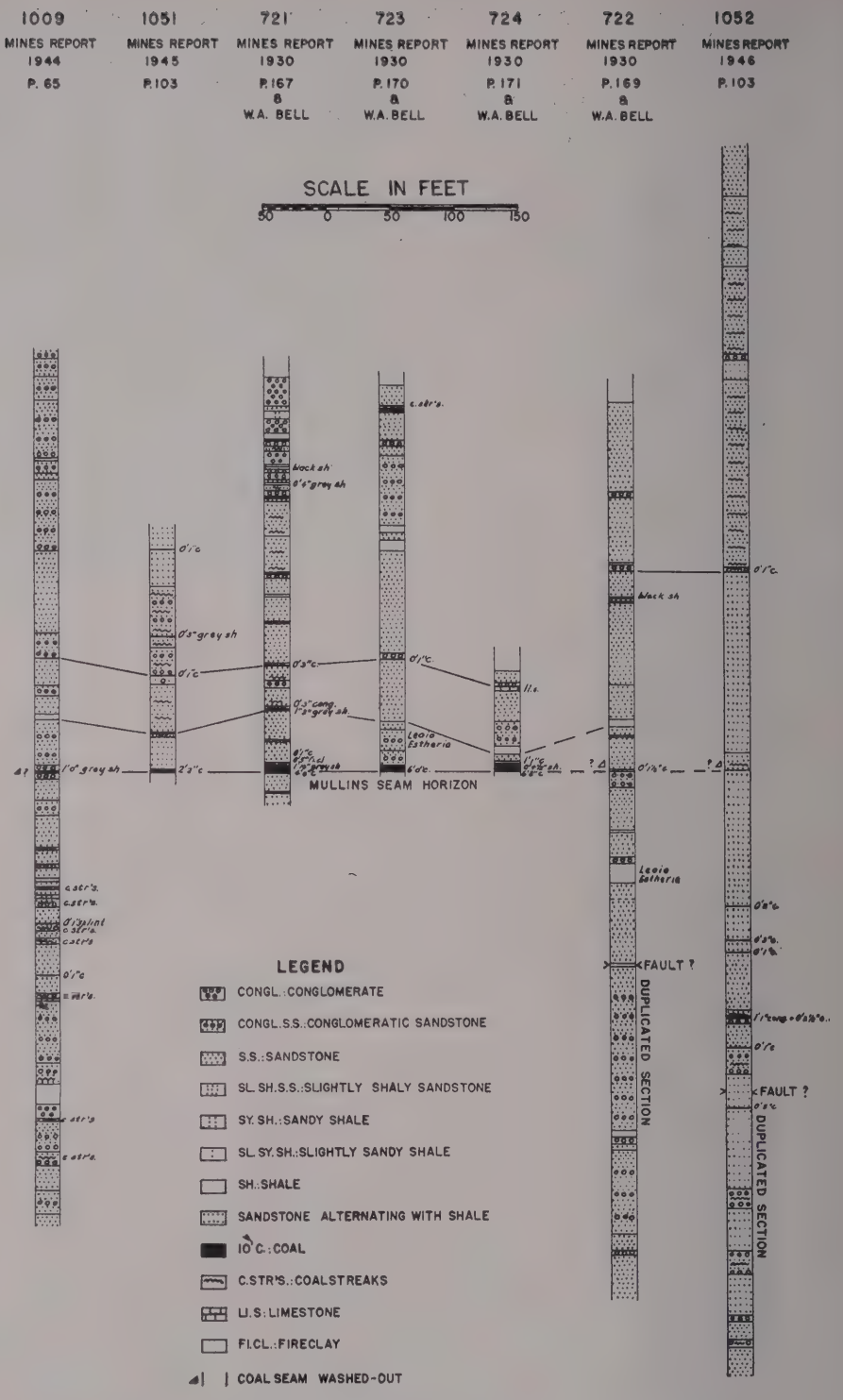


Figure 5.—Columnar sections of boreholes at North Sydney.

The fossil Morien river, which is held responsible for the deposition of the strata, grouped together under the same name, would have had to migrate northward in order to enable the swamps to enlarge, ascending in the stratigraphic column. As illustrated by the Phalen seam, in which two salient stages of river development were traced by conspicuous partings, the Morien river meandered back and forth over its flood-plain, at one time in the present submarine area (*cf.* parting in lower part of Phalen seam as struck by Dominion Nos. 18, 16, 4, 6, and North Atlantic collieries), at others on the present land area (*cf.* the Bridgeport-Lingan parting in top part of Phalen seam.)

It should be stressed that, as far as mineability is concerned, the effects of the lower Phalen wash-out

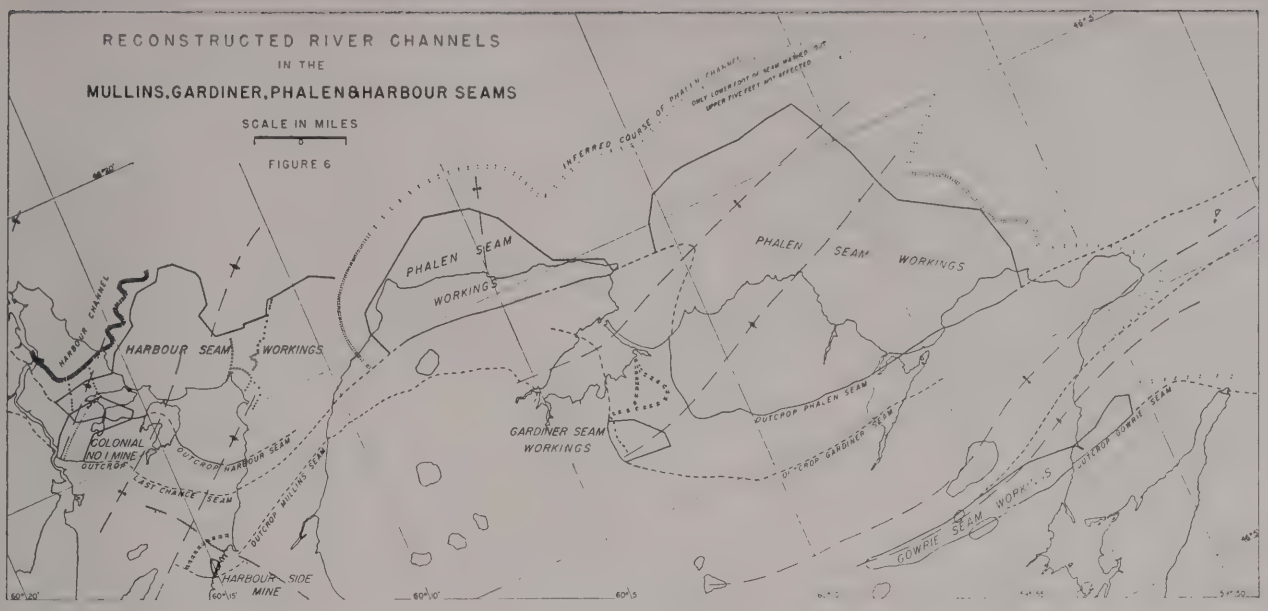


Figure 6.

are different in the New Waterford and in the Glace Bay district. East of Bridgeport and Lingan, a rider seam is joined with the main seam. The average thickness of the Phalen seam in this district is 7½ feet, the lower foot of which is lost on the east side of Dominion Nos. 4 and 6 collieries due to the presence of the lower Phalen wash-out (see 13, Figure 3). However, this river did not affect the upper 5-feet-thick branch, which constitutes a mineable seam of good quality coal. In the New Waterford district, the rider seam is not joined with the main seam, which therefore averages only 5½ feet. On the west side of this district, namely in Dominion Nos. 18 and 16 collieries, the lower Phalen wash-out separates a 1½-feet-thick lower branch and a 2½- to 3-ft.-thick upper branch, which is not mined.

A frequency curve of swamp development in the Morien group undoubtedly would show maxima in the upper part or *Ptychocarpus unitus* zone, in which time interval the most attractive seams were deposited. Apparently, during this period, river interference with vegetation was least. What remains of the seams after erosional truncation may therefore amount to a cross-section exhibiting upward enlargement of the mineable areas, but it should be clear that this is accidental and not a rule to be used for forecasting the productive zones of, for instance, the Hub and Lloyd Cove seams.

The Hub seam was mined in a small area in the Glace Bay district, where it split in easterly directions. Its equivalent in the New Waterford district, called the Barrachois seam, contained too high sulphur and ash percentages to attract immediate mining interest.

The Lloyd Cove seam has been, and is, mined north of Florence. In northwesterly as well as southeasterly directions, the seam splits. This was one of the reasons why the Scotia No. 2, or Lloyd Cove, colliery at Cranberry Head was abandoned.

Therefore, the data presently available on these seams do not substantiate Gray's theory.

Strctural Analysis

The underground workings in the various seams exposed a number of small thrust faults which invariably strike northwest (13, p. 221). The larger of them, intersecting several seams, suggest an increase in throw and displacement with stratigraphic depth. For instance, the Morien thrust, which was encountered by the workings of the Gowrie and Dominion No. 22 Collieries proved to have a throw of 43 feet. The throw decreases southward to 28 feet near the outcrop of the Trunnelshed seam. Bore-hole *A*-2 (15, p. 38, No. 19) supposedly intersected the fault at −330 feet, the throw at this depth being 86 ft. 7 in. (see Figure 7). The increase in throw of the thrust amounts to one foot for every eight feet of strata. The observations are as yet insufficient to state whether or not this figure can be accepted as an average.

If thrust faulting proves to be more severe in the older strata, its effects should show up in the land area of the Sydney coalfield, where these sediments are now exposed by erosion. Unfortunately, much is yet to be learned about the structure as exhibited by the distribution of the thin seams south of Sydney, but there are some indications of the presence of big faults.

At Macdonald lake, the Tracy seam has been mined and was found to be much disturbed. It is thought that a northwest trending thrust offsets the Tracy at this locality, throwing the seam back possibly as far as Black brook. This would correspond with a throw of 400 feet, a figure obtained from conditions found in a string of bore-holes along the old Cow Bay road (see Figure 8). The displacement along this thrust fault may be as much as 2,200 feet.

At North Bar, the Mullins seam is intersected by a fault, which is thought to be a thrust with a throw of 100 feet (see Figures 4 and 5). Correlation of the bore-holes supplying the evidence is difficult, due to the occurrence of a fossil river in this area which caused the seam to be absent in some of the holes. Assuming the strike of the fault to coincide with the general northwesterly trend of the thrusts, its extensions in northwesterly and southeasterly directions can be traced. Bore-hole 1067 (N. S. Dept. of Mines, Ann. Rept. 1945, p. 110) did not intersect the coal, supposedly found by Anthony in his pit and bore-hole. Bore-hole 641 (N.S. Dept. of Mines, Ann. Rept. 1923, p. 146) struck "troubled ground" underlying highly dipping strata at 750 feet. If the seams mined in the Gannon and Elkins slopes to the south of this hole were the same, the throw of the fault passing between the slopes would amount to

100 feet. Southeast of North Bar, this fault would pass through the South arm of Sydney harbour. After re-entering the land area, it is thought to run east of the city reservoir and west of Boutellier lake. Northeast of the first locality is the coal occurrence at Hall's farm. Here, three feet of coal have been mined, which were reported to be twisted and rapidly petering out. East of Morrison lake, the sudden change in strike of the seam outcroppings is thought to be more conveniently explained by the presence of a fault than by an anticline (16, Map 361*A*). In its southeastern extension, this fault may follow Spencer brook and eventually connect with the Bateson fault (see Figure 1). As indicated on map 361*A*, this fault supposedly follows the contact of Morien and early Palaeozoic strata. However, a Watt vertical variometer survey along McDonald road failed to show magnetic anomalies sufficiently large to indicate the presence of a fault (24, p. 6).

In this area, the Bateston fault does not adhere to the general northwesterly trend. Deviations from this strike may have occurred wherever the thrusts ran into the pre-Carboniferous buttresses. For this reason, the George River fault is thought to connect with the Mountain fault at New Campbellton (16, Map 360*A*). On Boularderie island, the connecting link may follow the general northwesterly trend. Evidence of this may be afforded by the faults shown intersecting the shore between Saunders cove and Groves point (but interpreted differently by Bell).

Furthermore, the change in strike along the shore near Big Bras d'Or offers additional support for the presence of a fault.

Near Cape Dauphin, the Mountain fault forks into a west and an east branch, the latter of which rapidly loses momentum and is thought to peter out within a mile, going along the strike. The major branch swings westerly. It may regain its northwesterly trend once more and connect with the fault, which intersects the Lower Windsor at North shore.

The smaller thrusts observed in the underground workings dip to the northeast and suggest a compression from that direction. If the Bateston and Mountain faults are similar faults, their history has to be more complicated. At present, their high sides are on the south, respectively west of these faults. Bell (1, p. 8) therefore considered these faults to be downthrusts of Morien strata against older rocks. Contrary to this opinion, it is thought that both faults were initiated during the Arbuckle phase of orogenetic movements as low-angle thrusts from the northeast (13, Figure 5). During the Appalachian phase they were folded, and subsequently, during the final emergence of the basin of deposition, the movements along parts of these fault planes were reversed. In the case of the Mountain fault, this brought Morien strata in contact with the crystalline rocks of Kelly mountain; in the case of the Bateston fault, Morien strata came in contact with Lower Carboniferous and early Palaeozoic rocks. Support for this is afforded by the dip of the fault plane, which is 20° to the bedding.

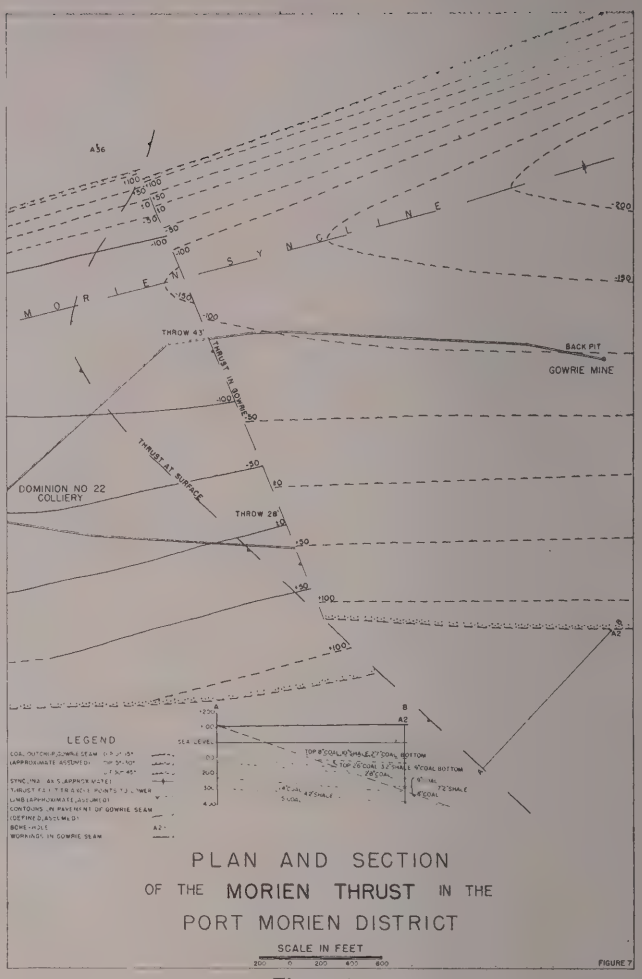


Figure 7.

The Mississippian beds at North shore can be followed, although not continuously, up to Cape North. Significance is attached to the fact that the strike of the Morien strata on Hertford and Ciboux islands is almost parallel to that of the forementioned Mississippian beds. At Cape Dauphin, the Canso follows the Windsor concordantly and a concordant relationship also seems present between the Canso and the overlying beds of the Morien group (see Figure 3). According to Bell (16, Map 359*A*, text), "the upper contact, however, is definitely a surface of erosion and it seems probable that a time break is represented at the lower contact also". In view of these concordant relationships it is thought that the Morien strata may follow the Mississippian northward, although at a considerable distance offshore.

The fold pattern of the Sydney coalfield has been interpreted before (13, p. 221). For speculation as to its seaward extension, the hydrographic charts offer some evidence (see Figure 9). On the land area, the Cap Percé anticline is a well defined major structure, trending east-west north of Birchgrove and east-northeast east of Sand lake. At Cap Percé, the trend swings again more to the east and this seemingly continues off-shore for a distance of 30 miles. The Morien and Glace Bay synclines as well as the Bridgeport anticline seem to lose their importance in a seaward direction. The synclines appear to widen and are less pronounced in the submarine area than on the land. The Bridgeport anticline is well defined at the head of the Bridgeport basin, but broadens in a seaward direction. To the north, a few indications of ready exist for the presence of another syncline (13, p. 221). The submarine topography suggests that this Lingan syncline may shape into a major fold, extending over a distance of 30 miles. The same conclusions hold for the New Waterford anticline, not indicated in the land area, but occurring in the submarine workings of the Harbour and Phalen seams. Less well defined by submarine contours are two seemingly synclinal folds separated by a possible anticlinal axis which, from east to west, may reflect the Florence syncline, the McCreadyville anticline, and the New

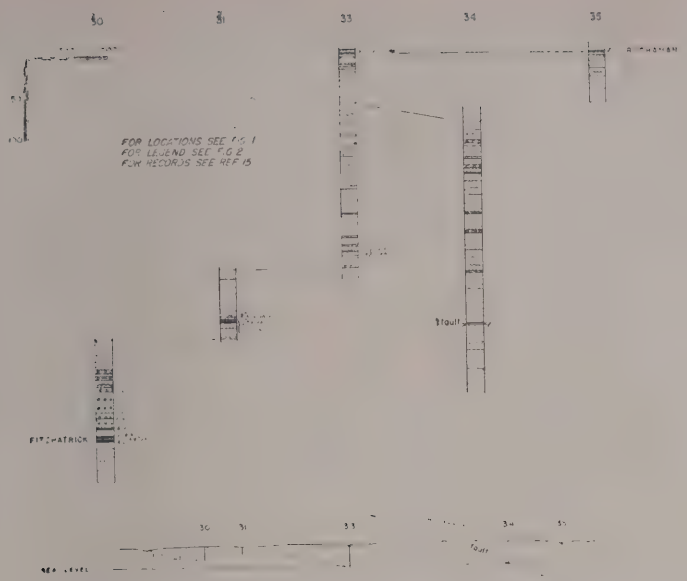


Figure 8.—Correlation of boreholes on Cow Bay road.

Figure 9.—Inferred submarine extension of the fold pattern in the Sydney coalfield.
(See Refs. 4 and 5)

Campbellton syncline (13, p. 221).

If the submarine topography has been interpreted correctly, a continuous plunge of the fold axes must be postulated. This structural pattern comes to an abrupt end where it is cut off by the inferred fault running through Cabot strait (5). This conclusion is not in agreement with Gray (8, p. 3) or with Bell (1, p. 101), who assumed that, within a distance of 30 miles offshore, "the strata might be expected to pass beyond the structural bottom of the basin and to dip landward".

Unexploited Areas

Three areas warrant a closer examination, because they are situated near, or partly even on, the land and so far have not been mined. They are, from west to east, the areas of Great Bras d'Or, Lingan, and Morien Bay.

Great Bras d'Or Area

The analysis of the submarine contours suggested that the New Campbellton syncline may continue seaward. In New Campbellton this syncline is deformed due to its nearness to the Mountain fault, of which the youngest movements are thought to have occurred after the folding. That this syncline must be regarded as a major feature is demonstrated by its southward continuation, termed the Boularderie syncline (16, Map 360*A*). Consequently, the coal seams in it may be thicker than elsewhere (13, p. 219). That this does not necessarily affect their quality was proved by the Four-foot seam in New Campbellton. The coal of this seam has a very bad reputation because it is liable to spontaneous combustion. Since this seems to apply to most of the coal seams west of Sydney harbour (H. C. M. Gordon, oral communication), the interest in the seams in the area under consideration will be limited.

Very little is known about the Six-foot or Phalen seam in this district, the few exposures indicating that the seam is split into several branches (see page 294). The Four-foot or Backpit seam, where mined, proved to have a steady thickness of four feet. Its equivalent, the Blackrock seam, as exposed on the east shore of Great Bras d'Or, is only three feet thick. Thus it seems safe to assume an average thickness of 3½ feet for the submarine area.

The Harbour seam intersects the shore south of Table Rock. Here, it consists of several thin branches, the thickest of which is only 12 inches. In the New Campbellton district, prospecting for this seam has not taken place, although it may intersect the shore one mile south of Cape Dauphin.

The Stubbart seam attains mineable height on Boularderie island, northwest of McCreadyville. In the Coastal Coal colliery, five feet of coal were mined between 1920 and 1922. The seam gains height rapidly and is eight feet thick, south of Outer Table rock. If this condition persists in the submarine syncline, an excellent seam of coal, as far as quantity is concerned, has not yet been touched.

The Lloyd Cove seam has been, and is, mined north of Florence, where it attained a thickness of as much as eight feet. Going north, the seam splits into two branches, the upper one 2 to 2½ feet thick and the lower one 3 to 4 feet. At Alder point, they are separated by 14 feet of strata, increasing to 28 feet on the east shore of Boularderie island, where the seams are called the Upper and Lower Bonar. A decrease of 8½ feet of strata takes place crossing over to the north shore of the island. Should this decrease persist, a seam of six or more feet thick may be present in the submarine area.

The Point Aconi seam, the highest in the section, is exposed at two localities, *viz.*, Cranbery Head and Point Aconi. At the last mentioned point it is 3 ft. 2 in. thick.

Thus, the Great Bras d'Or area, a submarine district, situated between the Bird islands and the north shore of Boularderie island, may contain mineable quantities of coal in the Lloyd Cove and Stubbart seams. Of the other seams, overlying and underlying the ones mentioned, data do not permit the drawing of conclusions, nor does the quantity of coal in the thick seams allow speculation as to its quality.

Lingan Area

Between the submarine workings of Dominion Nos. 12 and 16 collieries in New Waterford and Nos. 1-*B* and 26 collieries in Glace Bay, an area was purposely left untouched for a future reserve. A major fold, termed the Lingan syncline, may originate in this area. Structurally, it is situated between the New Waterford and the Bridgeport anticlines. Recently, sliding phenomena, which are thought to accompany this synclinal fold, were observed in Dominion No. 26 colliery, on the

north flank of the Bridgeport anticline. On the west side of this colliery, over a distance of 750 feet downdip, the seam increases in thickness from six to nine feet. Its roof consists of a slickensided shale showing striation parallel to the dip. A calcite film occurs on the slip-plane between the shale and the coal. Another feature attributed to sliding is the structural roll, striking N.70°E, which was observed on No. 2 North levels in this colliery. The sandy-shale roof rock appears to have plowed into the underlying coal, replacing the seam from the roof down to the pavement. The stone body is asymmetric, its uphill or south flank dipping 13° to the bedding and the north flank dipping 24°.

Palaeogeographic conditions also yield support for the presence of a syncline in the Lingan area, since the Phalen and Harbour seams were found to thicken in approaching the district.

Of the individual seams in the reserved area, very little is known about the Gardiner and Emery. Data obtained on both shores of Indian bay do not warrant optimism with respect to their mineability in the submarine area. On account of its even thickness of five feet on the east side of Dominion Nos. 15 and 16 collieries and of seven feet on the west side of Dominion No. 1-*B* colliery, the Phalen seam in the reserved area could be expected to average six feet. However, it is possible that a split occurs one mile offshore, greatly reducing or even spoiling the mineability of the seam in part of the district under consideration (see Figure 10). This split caused termination of the workings on the west side of Dominion Nos. 18 and 16 collieries. Indications are that the fossil river responsible for the split swings eastward and may eventually cut off the coal to the dip of the present Dominion No. 16 colliery. On the east side of this mine, thin partings at the corresponding height in the seam already suggest that the initial stages of the split condition are in existence in the area that borders the reserved district (11, p. 11). Since similar indications have been obtained on the north side of Dominion 1-*B* colliery, it is thought that, one mile or more off-shore, in the Lingan reserve, the Phalen seam may be split in a lower and an upper branch, respectively two and three feet thick.

A second parting, playing an important role in this area, occurs at the top of the seam. This split was

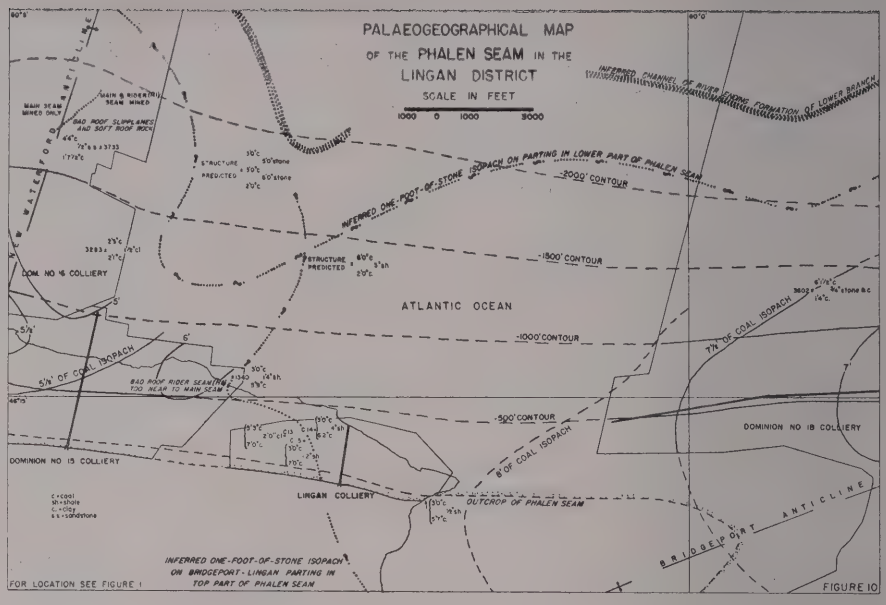

PALAEOGEOGRAPHICAL MAP
OF THE PHALEN SEAM IN THE
LINGAN DISTRICT
SCALE IN FEET
1000 0 1000 3000
INFERRED CHANNEL OF RIVER CUTTING FORMATION OF LOWER BRANCH
MAIN SEAM MINED ONLY
BAD ROOF, SLIPPY AND/OR SOFT ROOF ROCK
STRUCTURE PREDICTED
ISOPACH ON PARTING IN LOWER PART OF PHALEN SEAM
-2000' CONTOUR
-1500' CONTOUR
DOM. NO 16 COLLIERY
ATLANTIC OCEAN
-1000' CONTOUR
7/8" OF COAL ISOPACH
5/8" OF COAL ISOPACH
BAD ROOF RIDER SEAM TOO NEAR TO MAIN SEAM
-500' CONTOUR
DOMINION NO 18 COLLIERY
8' OF COAL ISOPACH
DOMINION NO 15 COLLIERY
LINGAN COLLIERY
OUTCROP OF PHALEN SEAM
BRIDGEPORT ANTICLINE
INFERRED ONE-FOOT-OF-STONE ISOPACH ON BRIDGEPORT-LINGAN PARTING IN TOP PART OF PHALEN SEAM
FOR LOCATION SEE FIGURE 1
FIGURE 10

Figure 10.

first observed in the abandoned Bridgeport colliery. Here, a three-feet-thick rider seam was found to overlie the six-feet-thick main seam. Strata, amounting to a maximum of 28 feet (3, p. 32) separate the two seams near the outcrop, but in easterly directions they gradually decrease, the rider and main seams eventually joining to one seam, which then attains a thickness of 7½ feet. The one-foot-of-stone isopach between the rider and the main seam can be traced to the abandoned Lingan colliery and the east side of Dominion No. 15 colliery (7, p. 671). It then swings northerly, judging by the presence of the rider seam on the west side of Dominion No. 1-*B* colliery (height of coal 7 feet, rider seam included) and its absence on the east side of Dominion No. 16 colliery (height of coal 5 feet).

In Dominion No. 15 colliery, bad roof conditions resulted from the nearness of the rider seam to the main seam. Similar difficulties may arise in the Lingan reserve in a zone on the west side of the one-foot-of-stone isopach. Eventually, the submarine district, also, will probably be influenced by the bad roof conditions, recently existing on the east side of Dominion No. 16 colliery. Here, a relatively soft roof rock, consisting of slightly sandy shale, is intersected by slip-planes, striking parallel to the wall faces. Locally, this condition caused five feet of stone to fall with the extracted coal. After studying the evidence, it was predicted and later confirmed that the slip-planes were connected with the crest zone of the New Waterford anticline and therefore should decrease in frequency of occurrence toward the flanks of the fold. Since the anticlinal axis is thought to trend eventually into the reserved area, bad roof conditions are likely to occur there also in the anticlinal crest zone.

The Backpit seam, although measuring 4 feet on both shores of Indian bay, decreases in thickness in a seaward direction, as indicated by submarine bore-holes in Dominion Nos. 1-*B* and 16 collieries.

The Harbour seam may be expected to have an average thickness of 5½ feet in the reserved area. Locally, this figure may be higher, for in Dominion No. 1-*B* colliery the seam was found to increase in thickness down-dip on the north flank of the Bridgeport anticline. On the other hand, the bottom part of the seam, one foot or less thick, may be dirty. This condition occurs on the west side of Dominion No. 12 colliery and is indicated also on the east side of this mine by a few sections. If persistent, it may also affect the submarine reserve.

The Hub is the highest seam in this area of which data, albeit only a few, are available. The stone drift from Dominion No. 14 colliery to this seam encountered almost six feet of coal. The section on the north shore of New Waterford, as reported by Robb (18, p. 224), consists of 3 ft. 0 in. (top) coal, 3 ft. 3 in. fireclay and shale, and 5 ft. 0 in. (bottom) coal. This section seems to indicate that conditions are reversed as compared with those in the abandoned Dominion No. 7 — or Hub — colliery in Glace Bay. Here, the lower branch averages 3 ft. 3½ in., whereas the upper branch is 5 feet thick. In view of this uncer-

tain correlation, it seems best to use a thickness not exceeding 4½ feet for purposes of estimation.

The Lingan reserve is thus thought to contain mineable quantities of coal in the Hub and Harbour seams. A split may occur in the Phalen seam, rendering it uneconomical to work this coal in the more distant offshore parts of the area.

Morien Bay Area

Structurally, the Morien Bay district forms the submarine continuation of the Port Morien syncline. This syncline may eventually disappear if the suggested interpretation of the submarine topography is correct. Two thrust faults observed along the shore north of Port Morien may intersect this area. The most northerly of these faults is half a mile north of Long Beach. It probably forms the continuation of the thrusts observed in the underground workings of Dominion Nos. 4 and 24 collieries. In the Emergency travelling road in Caledonia (No. 4) colliery, a throw of 8 feet and a displacement of 25 feet were measured. In the underlying Emery seam, the throw was the same, but the displacement had increased to 35 feet. In the shore section, these data could not be obtained, due to the lack of index beds on both sides of the fault.

Supposedly also a thrust is the fault observed at Milbrook cove. It is thought to intersect bore-hole *A*24 (see Figure 11), thus explaining, more satisfactorily than without a fault, the absence of the Trunnelshed seam in this hole. The throw as calculated amounted to 70 feet.

On account of the folded nature of these thrusts (13, p. 224) their extensions in the submarine areas are difficult to forecast. There are no records indicating that the North Atlantic colliery was intersected by faults; consequently, the Milbrook Cove thrust is thought to curve easterly in such a manner that it would miss those abandoned workings.

Mining in the submarine district has been carried on only in the abandoned North Atlantic colliery, in the Gowrie or Phalen seam. This seam averaged 5 feet in thickness, of which the top 3 inches consisted of roof splint. A parting, 8 inches above the pavement, amounting to 1 inch or less in the upper workings, increased down-dip to 6 inches, rendering the lower foot of the seam valueless. This parting occurs at the same height above the pavement as the parting on the east side of Dominion Nos. 4 and 6 collieries (13, Figure 3). Apparently, the fossil river responsible for this split also affected the Morien Bay district during the time interval in which the split originated.

In the North Atlantic colliery, that part of the seam overlying the split averages 3 ft. 10 in. Another parting, roughly in the centre of the seam, may be expected to increase in thickness in easterly directions, judging by the split and dirty condition of the seam exposed at Cape Morien. Consequently, considerable uncertainty exists as to the mineability of the Gowrie seam in the remaining parts of the submarine area. An analysis of a sample taken in the North Atlantic colliery showed the seam to have high sulphur and ash contents (15, p. 67).

The Blockhouse or Harbour seam is up to 9 feet thick in the relatively small land area. The presence of the Milbrook Cove fault and the required rock cover of 180 feet coupled with the 2°30′ pitch of the synclinal axis (15, p. 67), render

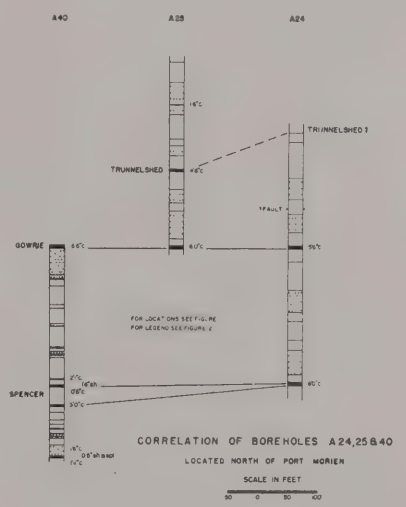


Figure 11.

the seam inaccessible within a mile off-shore. In the abandoned Blockhouse colliery, clay-gashes occurred on the south flank of the syncline (19). These tension cracks may also be present in the axial zone of the submarine syncline (13, p. 223).

The only other seam, which could contain mineable quantities of coal is the Spencer or Emery. An account of the available data on this seam has been given by Bell (15, p. 63), who concluded that the seam may be 5 feet thick under the ocean; but that its quality could be poor.

The Port Morien submarine district may thus contain mineable quantities of coal in the Spencer, Gowrie, and Blockhouse seams, but only the last seam looks promising, when an extrapolation of the available information on the land area is justified. Except by means of expensive rock tunnelling, this seam cannot be reached from the shore, but would have to be mined one mile or more off-shore throngh cross-measure drifts from a development in the Gowrie.

Summary and Conclusions

The Sydney coalfield has always been regarded as a field in which the limitations accompanying mining of coal at great distances from shore would be reached before the edges of the field were approached. The symmetry or asymmetry of the drainage system in the individual seams will determine in each case the extent of the potential reserves. For instance, on the northwest side of Florence colliery, the Harbour seam is cut off by a fossil river. Except for bore-hole *D*7 near Alder point, where 6 ft. 6 in. coal was reported, the Harbour seam on Boularderie island has not, so far, been found to be mineable. The Phalen seam shows a partial recovery on the east shore of Sydney harbour and possibly in the Colonial No. 1 colliery at Little Bras d'Or, but boreholes on Boularderie island do not substantiate any further optimism. Also, this seam cannot be depended upon to be free of splits in the unexploited areas. The Bridgeport-Lingan parting in the top part of the Phalen seam peters out in eastern directions. Consequently, here an extensive body of coal that has been mined ever since the early days of the Sydney coalfield occurs on the seaward side of a channel.

The two Phalen channels, the lower one of which was traced on the west side of Dominion Nos. 16 and 18 collieries and on the east side of Nos. 4, 6, and North Atlantic, the upper one occurring at Bridgeport and Lingan, are but two stages in the development of a big river. This Morien river deposited its sediments on a subsiding flood plain, thereby from time to time and from place to place interfering with swamp development. The overall course appears to be roughly parallel to the present coastline, its bends swinging seaward in anticlinal areas and landward in the synclines. The locations of the two Phalen channels suggest that there was no general northward migration of the Morien river, an essential condition for Gray's theory of progressive upward enlargement of the mineable areas in the seam (8, p. 5). More-

over, present data on the Hub and Lloyd Cove seams do not substantiate his ideas.

It is thought that the Morien river may represent an arm of the Pennsylvanian St. Lawrence River delta. Schuchert (20, p. 706) indicates that the New Brunswick geanticline separating the St. Lawrence and Acadian geosynclines was first submerged during the deposition of the Windsor, *i.e.*, Mississippian. It is thought that, except possibly for a few monadnocks, this geanticline maintained a low relief during the Pennsylvanian — a period of extensive flood-plain deposition. The sediments deposited were again partly eroded, causing a thorough sorting of the components. As a result, coarse conglomerates at the base of the Morien series — *e.g.*, those occurring on Ciboux island — may have been derived from underlying Canso deposits cut by the river channel. However, the main process was one of accumulation on a subsiding flood-plain, the river meandering over it.

There is a start and an end to the flood-plain development. As far as the Sydney coalfield is concerned, fluviatile sedimentation took place in Morien time. Now and again, periods of excessive sedimentation alternated with those of extensive swamp development. Orogenetic movements in the hinterland may be responsible for this variation (13, p. 220).

From a mining point of view, the first important period of swamp development is reflected by the Tracy seam. The correlation of the stratigraphic columns in Inverness and Sydney suggests that the Point Aconi seam may represent the highest seam in the Sydney field.

Although the seams may deteriorate in a seaward direction, their time horizons, represented by clastic sediments, carry on in that direction. The submarine topography implies that the fold pattern extends offshore for considerable distances, until it is finally cut off by the Cabot Strait fault, which trends northwesterly, *i.e.*, parallel to the known thrusts of the Sydney coalfield. The submarine contours do not suggest an eventual landward pitch of the strata.

At present, mining progresses in northerly and easterly directions. Since the thrust faults trend northwesterly, the east-going mining operations should encounter them first. Their throw and displacement seems to be less in the younger than in the older beds. The lower strata show signs of severe faulting, but since mining interests are mainly focussed on the seams which are higher in the stratigraphic column, it is not thought that thrusting will interfere seriously with future operations.

In conclusion, the shape of the Sydney coalfield can be defined as a huge triangle. Its base — presumably the Cabot Strait fault — strikes northwesterly; one side trends east-west, almost coinciding with the latitude of 46°00′, and the other trends northeasterly, coinciding with the Mountain fault at New Campbellton. Not all the seams enclosed in this area have economic importance. The mineable seams seemingly occur between the Tracy at the base and the Point Aconi at the top. Their lateral extent will vary, dependent upon the fossil drainage in each seam.

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MINING SOCIETY OF NOVA SCOTIA
DONALD J. MACNEIL, *President* S. C. MIFFLEN, *Secretary*

Coal Preparation Plant of Old Sydney Collieries, Limited, Sydney Mines, C. B.

By J. A. RUSSELL *

(*Annual Meeting, Mining Society of Nova Scotia, Sydney, N.S., July,* 1952)
(*Transactions, Volume LV,* 1952, *pp.* 213-216)

UNTIL the last few years, the collieries of the Dominion Coal Company, Limited, while highly mechanized as to the cutting and transportation of the coal, still relied on the hand-loading of coal from the seam onto face conveyors. In these circumstances it was possible to assure that no large proportion of extraneous materials would be loaded with the coal, and the product could be quite well cleaned for the market by screening and hand-picking.

Within the past few years, however, the Company has introduced an intensive programme of mechanization which will result in the virtual elimination of hand-loading of coal. It has been the experience of other producers that mechanical loading inevitably leads to a very large increase in the amount of impurities in the product. This is occasioned by the fact that, at least up to the present, all types of loading machines have relied on the coal being broken from the seam by undercutting and shooting. In this operation, a certain amount of roof and pavement material inevitably becomes mixed with the coal and, since selective loading is impossible with a loading machine, all this foreign material is loaded out with the product.

This condition does not necessarily apply to the *DOSCO* continuous miner which, under reasonable conditions of roof and pavement, can load clean coal, since it does not rely on explosives; but in spite of this, the Dominion Coal Company has accepted the principle that a highly mechanized coal operation requires the mechanization of the subsequent cleaning processes.

With this in mind, it was decided that the first modern coal preparation plant should be installed in the Sydney Mines area to clean mechanically the outputs of the mines operated by Old Sydney Collieries, Limited, and work on the design of the plant was commenced in the Company's Engineering Department at the beginning of 1950.

The plant was specified to be capable of handling a total of 4,900 net tons in sixteen hours of operation, 3,600 tons from Princess colliery and 1,300 tons from Florence colliery. In making up the flow-sheet, a figure of 325 tons per hour of raw coal was used, giving a theoretical capacity of 5,200 tons in sixteen hours of uninterrupted operation.

Careful float-and-sink tests of the coal in all sizes between 1/8 inch and 6 inches were made in order to determine just what the cleaning problem would be and how it could be most economically dealt with. Figures 1 and 2 are extracts from the essential portions of the washability curves lying between specific gravities 1.30 and 1.80, showing the theoretically possible percentage recovery and the theoretical ash content of the cleaned coal resulting from washing at the different gravities.

It will be seen from these curves that both coals are quite easy to wash, and it was decided that it should be possible to obtain satisfactory cleaning by washing all sizes together, *i.e.*, by washing all sizes, between 1/4 inch and 6 inches, together in a single washer of the Baum type, without any necessity for elaborate plant for middlings recovery.

On the basis of tenders received, a contract was awarded for a Norton-type automatic jig, to be supplied by the McNally-Pittsburg Manufacturing Corporation. This Company guarantees that, of the Florence coal, not more than 0.93 per cent by weight of materials sinking at 1.60 specific gravity shall report to the clean coal, and that not more than 0.86 per cent of the feed tonnage less than 1.60 specific gravity shall be found in the rejects. Corresponding figures for the Princess coal are 0.91 per cent sinks in the clean coal, and 0.92 per cent floats in the dirty coal. This corresponds to a recovery of 93.19 per cent of the Florence coal, and 93.8 per cent of the Princess coal.

Figure 3 is the basic flow-sheet which was established for the plant. In laying this down, we had in mind that quite a number of the older plants in the United States have had to be rebuilt because their refuse disposal systems were not adequate to handle the considerably larger tonnage of refuse resulting from mechanization. For this reason all the equipment in the refuse side of the plant was designed to handle a possible 25 per cent of the input tonnage as refuse, a figure which we sincerely hope will never be reached. This feature of the design will explain some discrepancies in tonnages and percentages which would be apparent from a close scrutiny of the flow-sheet and the washability curves.

It was also decided that, since the introduction of very large lumps into a jig upsets its operation, and

*Chief Mechanical Engineer, Dominion Coal Company, Limited, Sydney, N.S.

since very large lumps of coal are no longer a premium product, all the coal would be reduced to a maximum of six inches before cleaning.

As mentioned earlier, this is the first up-to-date plant to be built in Cape Breton, and, since others are to follow, we have taken the opportunity to incorporate in this plant as many different types and makes of equipment as we could, with the idea of learning all about them before we incorporated them into other, and possibly larger, plants. I mention this because it explains what might otherwise appear to be a haphazard mixture of equipment in the plant.

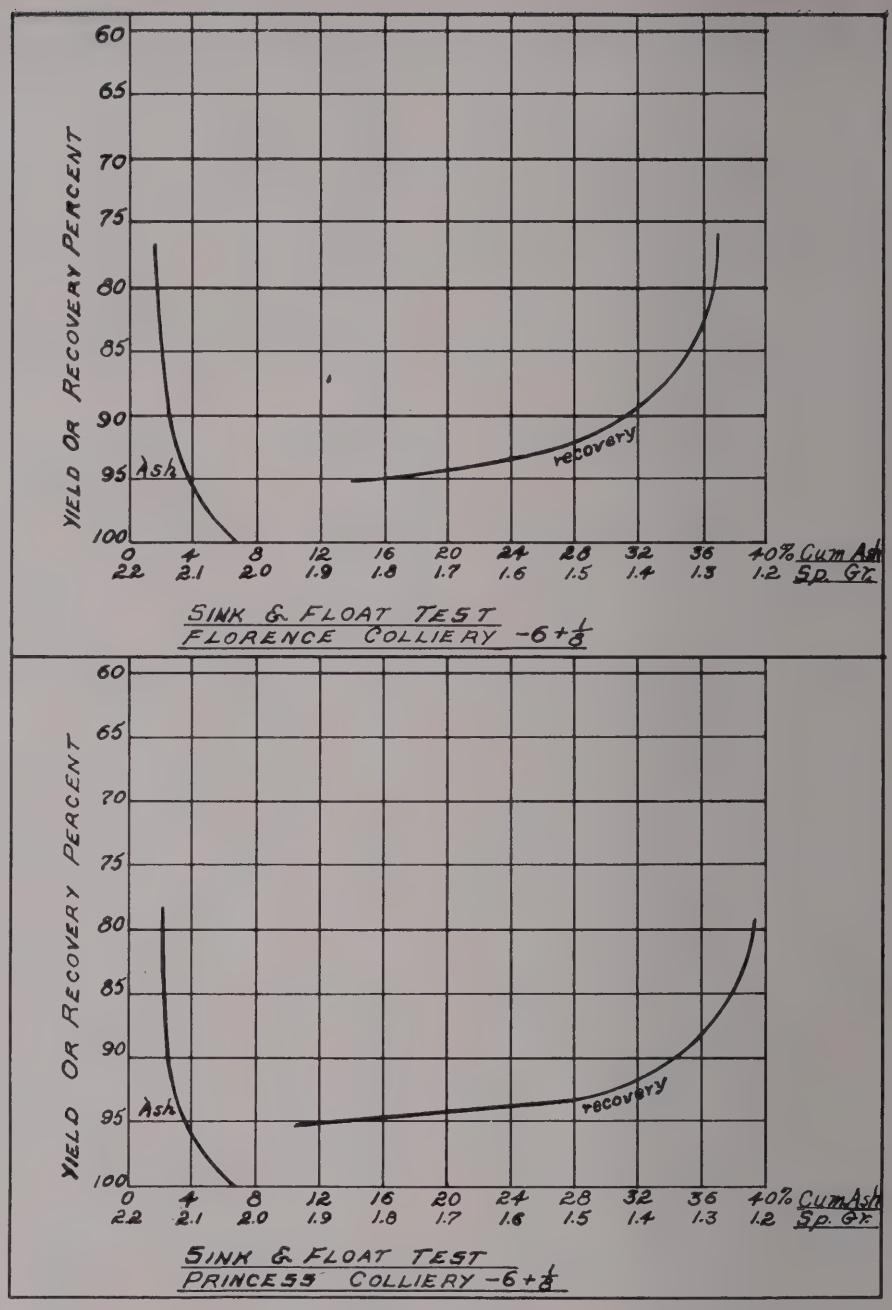
Coal will be received from the collieries in 50-ton frog-mouth hopper cars, which will be unloaded over a double compartment track hopper large enough to handle the modern, large three-hopper cars without shifting. Final clearing out of the cars will be facilitated by the use of a Robins car shake-out. From the track hopper, the run-of-mine coal will be fed by dual reciprocating feeders to one 42-inch rubber belt conveyor, which will transport it to a Bradford breaker.

Within the next two years or so, a belt conveyor tunnel from Princess colliery will be completed, and Princess coal will then be transported directly by belt conveyor to the surface and will be transferred directly to No. 1 plant conveyor without going through the track hopper at all.

The 9 ft. by 16 ft. Bradford breaker will reduce the coal to a maximum of six inches. A Bradford breaker was chosen in preference to a crusher primarily because it was felt that, since all coal below six inches will very rapidly pass through breaker screens, the reduction in size would be obtained with a minimum of breakage.

The Bradford breaker also has the decided advantage that it will throw out large pieces of hard stone before they even enter the plant, and it is calculated that a possible sixteen tons per hour of very objectionable material will be thus eliminated. This refuse will be carried by a 36-inch belt conveyor to a bin, where it will be loaded in trucks for final disposal.

The broken coal, which is now 6 inches by zero, will be transported from the Bradford breaker by a 36-inch belt conveyor, some 400 feet long, to the highest floor of the preparation plant proper. On this floor are installed three vibrating screens, each 5 ft. by 14 ft., a Ty-Rock, a



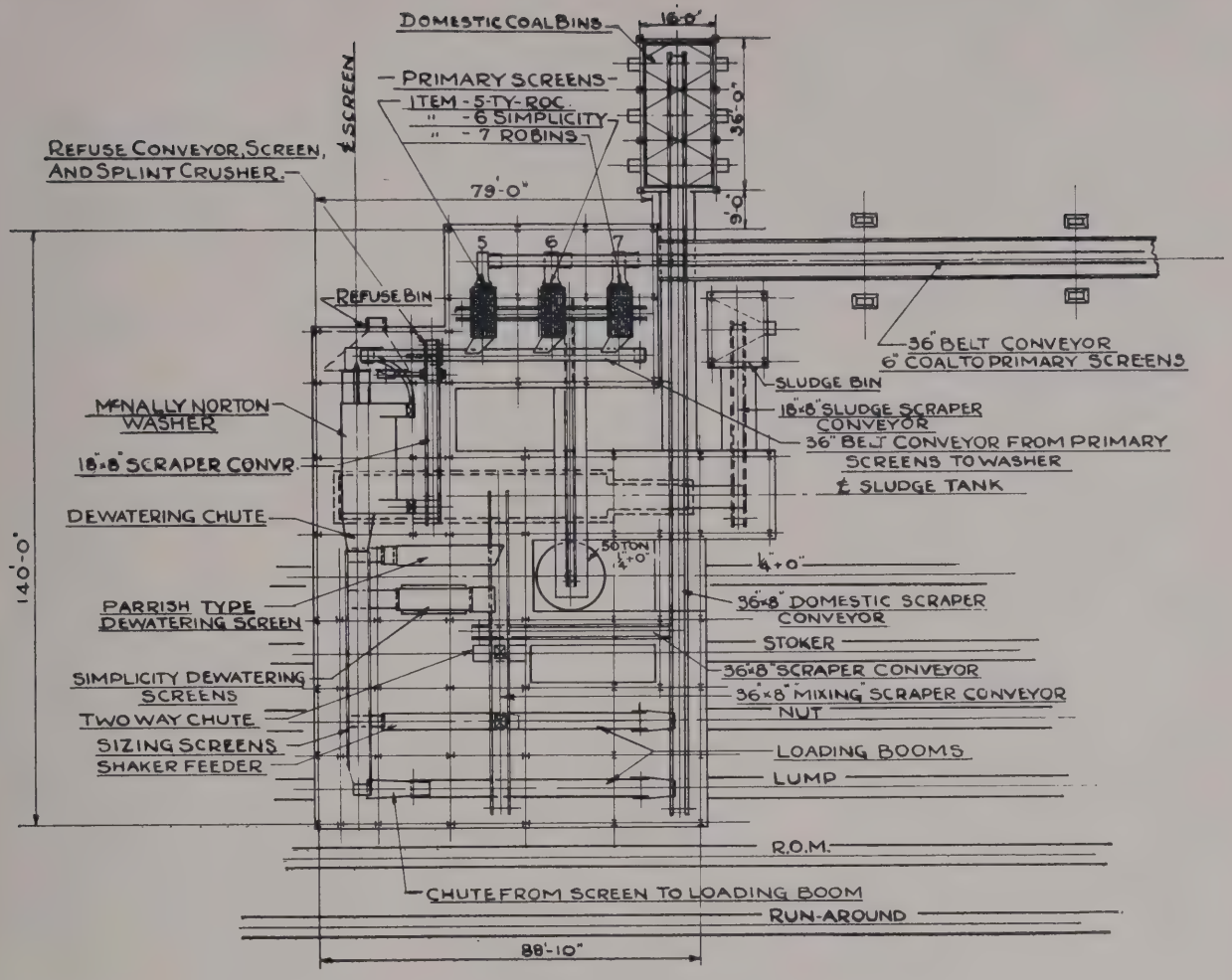
Figures 1 and 2.

Robins, and a Simplicity, each having a normal feed capacity of 100 tons per hour, with the ability to handle 150 tons per hour with reasonable efficiency in case of a breakdown.

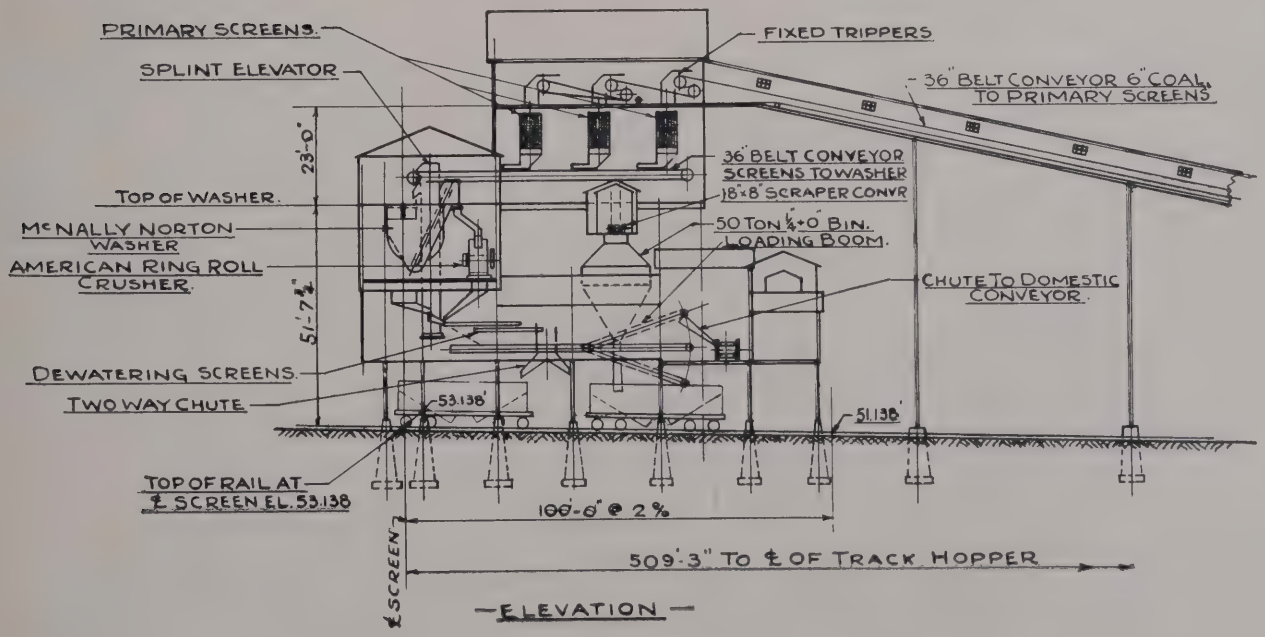
These screens remove the minus ¼-inch sizes, which are transported directly to a 50-ton storage bin, or which can be remixed with the coal after washing. The ¼ in. by 6 in. coal, now amounting to some 265 tons per hour, is transported by scraper conveyor from the primary screens directly to the McNally Norton washer.

Material evacuated by the primary refuse elevator is sent directly to a 25-ton refuse bin. That from the secondary elevator is screened through a 3 ft. by 6 ft. Simplicity vibrating screen with one-inch openings. Material passing through the screen is conveyed to the refuse bin; that passing over the screen goes through a ring roll crusher, which reduces it to one-inch, and is then returned to the jig, where coal which has been freed from adherent stone by the crushing will be recovered.

Two hundred and forty tons per hour of clean coal from the washer next passes to a pair of balanced pendulum hanger shaking screens carried on anti-friction bearings, a design of suspension originated by the Dominion Coal Company and used with considerable success for over ten years. These screens serve the functions of separating the coal into its market sizes and of carrying out the preliminary dewatering of the smaller sizes and the full de-



— COAL PREPARATION AND LOADING PLANT —
— GENERAL PLAN —



— ELEVATION —

— SYDNEY MINES WASHPLANT. —
— PLAN AND ELEVATION. —

Figure 4.

watering of the larger sizes.

Sizes which will be initially produced are:

0—¼ in. ..... Unwashed fines
¼ in—¾ in. ..... Washed stoker
¾ in—1¾ in. ..... Washed nut
1¾ in.— 6in. ..... Washed lump

The ¼ in. by ¾ in. size, after leaving the shaking screens, passes to two batteries of dewatering screens using ½ mm. wedge wire on the lower decks. One battery consists of two Simplicity vibrating screens, and the other of a balanced pair of Parrish-type, high-speed shaking screens, designed by our own Engineering Department. In designing the plant, sufficient space has been left so that, if it is found desirable, additional dewatering screens can be installed for the other sizes of coal.

After dewatering, the coal can be loaded in the above-mentioned sizes into railway cars, or can be re-mixed in almost any combination of sizes before loading into cars. The ¾ in. by 1¾ in. and 1¾ in. by 6 in. sizes are placed in the cars by scraper-type loading booms to prevent breakage; the ¼ in. by ¾ in. size is loaded through a bifurcated chute. All sizes except the zero by ¼ in. can be oil-sprayed to suit customers' requirements.

In close proximity to the plant there will be a battery of bins for the loading of coal into trucks supplying the local market, and a conveying system is provided so that the various sizes of coal may be transported into these bins directly from the wash plant.

The main reservoir for the water which is used in the washing process, and which is re-circulated through the plant, is a reinforced concrete tank of approximately 40,000 gallons capacity. From this, water is pumped to feed the jig, and to it all water recovered from the dewatering chutes and screens is returned.

The tank is of such size and design that all except the extremely fine sizes of the coal particles which are carried back with the water will sink to the bottom of the tank. In the tank bottom there is a very slowly moving chain scraper conveyor which, without unduly agitating the

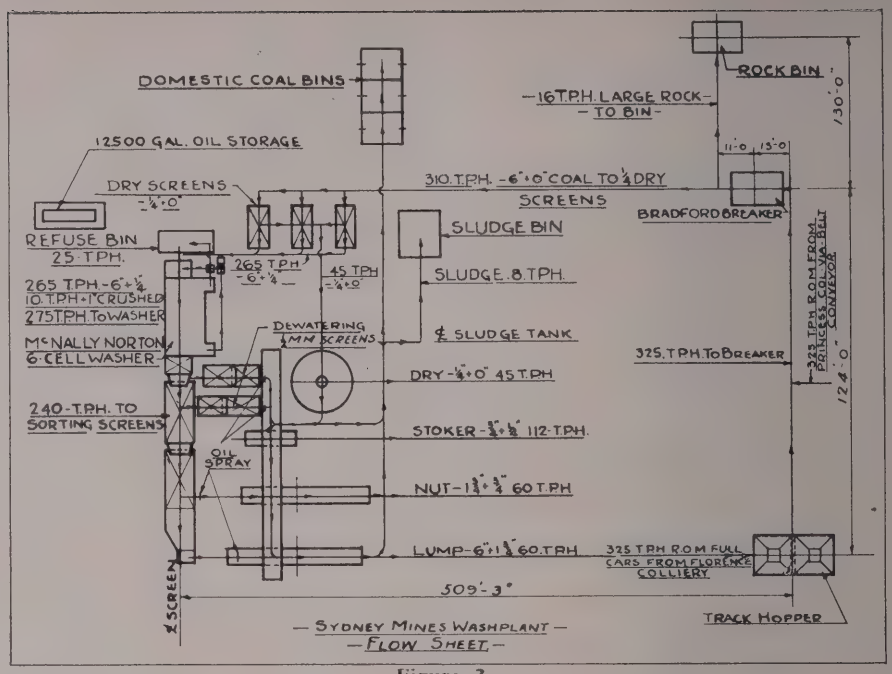


Figure 3.

water, will draw the slurry from the bottom of the tank up a draining incline or beach and deliver it to a cross conveyor.

This conveyor, which rises at a sharp angle, serves the purpose of further draining the slurry, which is finally deposited in a draining bin. From the bin it will be hauled by trucks to a stock pile, where it will be permitted to drain off still further. After a period in the draining stock pile, the slurry will be sufficiently dry to be recovered and used under our plant boilers.

The water is circulated through the plant by a 1,200 gallon per minute McNally Norton circulating

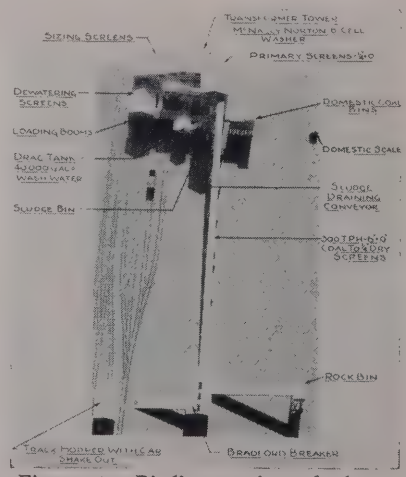


Figure 5.—Bird's eye view of plant.

pump. Two of these pumps are provided because the duty is quite severe and the failure of the pump can close down the whole plant.

The plant is housed in a structural steel building some 140 ft. by 90 ft. in plan, and about 90 feet high. Since the plant will operate in the winter, those portions of it where the coal is wet must be heated. Heated portions of the plant are covered on the sides with ¾-inch resin-bonded plywood, to which corrugated iron is nailed. Roofs are ¾-inch resin-bonded plywood covered with sheet roofing and an underfelt. Similar construction using tongue-and-groove sheathing has been very successful at other of our plants under severe weather conditions, but this is the first time we have used plywood.

The six tracks required for servicing the plant are laid with a falling grade of 2 per cent at the point where the cars are loaded and 1½ per cent elsewhere, so that cars may be handled with car retarders.

Equipment in the plant is all electrically operated, all motors being totally enclosed and all switches housed in a centrally located, dust-tight switch room. Current used is 550 volt, 3 phase, 60 cycle.

Figure 4 is a plan and elevation of the plant, and Figure 5 a bird's eye view of the plant as completed.

MINING SOCIETY OF NOVA SCOTIA
DONALD J. MACNEIL, *President* S. C. MIFFLEN, *Secretary*

The History of Basic Steel Manufacture At Sydney, Nova Scotia

By M. R. CAMPBELL *

(*Annual Meeting, Mining Society of Nova Scotia, Sydney, N.S., July,* 1952)

(*Transactions, Volume LV,* 1952, pp. 207-215)

INTRODUCTION

ON THE LAST DAY of December, 1951, a half-century of steel production was completed in the open hearth department of Dominion Iron & Steel, Limited, in Sydney, Nova Scotia.

During that period, almost every known variation of basic steel making practice was used in working out a steel making process which would overcome the difficult metallurgical problems imposed by the use of Cape Breton coal and Wabana ore.

This paper is presented as a brief review of these steel making practices, setting forth the fundamental metallurgical factors involved in their success or failure.

ESTABLISHING THE INDUSTRY

The story of steel production at Sydney is closely interwoven with that of coal. The steel industry owes its inception to a desire on the part of the Dominion Coal Company, which was incorporated in 1893, to improve its marketing position by increasing sales of slack coal and providing a year-round demand for its product. Accordingly, the Dominion Iron and Steel Company, Limited, was incorporated on March 30th, 1899, with Mr. H. M. Whitney — who was President of the Dominion Coal Company — as its first President. The new Company was given powers to manufacture and process iron and steel, and to mine all minerals, including coal.

A strong inducement to the industry was the bounty offered by the Dominion Government for the production of both iron and steel.

The *Lower Bed* of the immense Wabana iron ore deposit, which had then recently been discovered, had been purchased from the Nova Scotia Steel Company, and assured a practically inexhaustible supply of iron ore. Limestone deposits from which flux for the blast furnaces and open hearths could be supplied were known to exist in the vicinity of Sydney.

On June 20th, 1899, the Steel Company entered into an agreement with the Coal Company for the supply of sufficient coal for its operations, from seams which the Steel Company could designate.

On July 1st, 1899, the first sod was turned in the construction of an integrated steel works consisting of four blast furnaces, each of 250 tons capacity, ten 50-ton tilting open-hearth furnaces, four hundred Otto-Hoffman coke ovens, and a 35-inch blooming mill.

The new industry was begun in an atmosphere of great optimism. Mr. A. J. Moxham, the Vice-President and General Manager of the Steel Company, in an address delivered to the Canadian Manufacturers Association in Toronto on February 15th, 1901, predicted a splendid future for the industry. He stated that the raw materials for steel making could be assembled at Sydney at lower cost than anywhere else in the world.

Table I gives the composition of coal and ore as then reported by the management.

TABLE I.— COMPOSITION OF COAL AND IRON ORE

	RESERVE MINE	CALEDONIA MINE	DOMINION MINE
RAW COAL:			
Moisture:	1.45%	1.54%	1.21%
Volatile matter	32.45	30.86	31.89
Fixed carbon	60.45	62.91	61.49
Sulphur	1.64	1.50	1.56
Ash	5.65	4.69	5.41
WASHED COAL:			
Moisture	1.01	1.08	0.84
Volatile matter	32.99	33.92	37.86
Fixed carbon	62.21	61.69	62.60
Sulphur	1.11	1.07	1.17
Ash	3.79	3.31	4.50
RETORT COKE:			
Sulphur	0.91	0.78	1.01
Ash	6.07	5.38	6.24

	BEST	WORST
BELL ISLAND IRON ORE:		
Moisture	1.50%	2.40%
Total iron	54.43	51.84
Silica	9.34	13.00
Phosphorus	0.744	0.835
Sulphur	0.05	0.03

*Open Hearth Superintendent, Dominion Iron and Steel, Limited, Sydney, N.S.

Blast Furnace Operation

Blast furnace operation began in February, 1901, and two furnaces were shortly in operation producing foundry iron. This iron was sand cast at the furnaces and was made from a fifty-fifty mix of Wabana and foreign ores.

When open hearth steel making began on December 31st, 1901, the blast furnaces were converted to the production of basic iron, using all Wabana ore in the burden.

As was to be expected with burdening materials largely untried, serious operating difficulties were soon encountered at the blast furnaces. Actual analysis of both coal and ore fell far short of the published figures. The Wabana ore from the outcrop was contaminated with rock, and was unsized. The unwashed coal from which the coke was made was higher than desirable in sulphur and steadily increased in this element, so that in 1903 it was running as high as 2.75 per cent. The high-silica ore and high-sulphur coke necessitated extremely heavy lime burdening, and produced viscous slags which interfered with smooth furnace operations. These difficulties caused rapid deterioration of furnace linings, and culminated in the burning through of No. 1 furnace after a few weeks of operation.

Furthermore, it was found very difficult to hold the analysis of the iron produced within the range desired by the steel making furnaces. It varied from high-silicon, which required unusually large charges of limestone and ore in the open hearth furnaces, to low-silicon with high sulphur. The latter element was the most difficult to remove and the result was the production of considerable steel off in analysis.

The First Years of Steel Production by the Open Hearth Process

The open hearth furnaces were designed by H. H. Campbell of Steelton, Pennsylvania, for the production of steel from high molten metal charges. Mr. Campbell was the foremost furnace designer of his day and was retained as Metallurgical Engineer by several of the steel companies in the United States. He had built the first tilting furnace at Steelton in 1889. His tilting type of furnace gave ease of removal of the large volumes of slag produced, and proved in the long run to be well suited for the Sydney practice. These furnaces were basic lined and were among the largest basic furnaces in existence at that time.

Basic iron supplied for steel making by the blast furnaces at Sydney had the following average composition:

Carbon	4.00 per cent
Phosphorus	1.50 "
Sulphur	0.060 "
Manganese	0.20 "
Silicon	1.50 "

In the basic open hearth process, the elements silicon, carbon, and phosphorus are removed by the interaction of oxides of iron with these elements in the molten charge to form oxides. Lime from the limestone in the open hearth charge forms stable compounds with these oxides of phosphorus and silicon and with the sulphur. These remain in the slag. The carbon forms carbon monoxide, which burns and passes out the stack. Limestone, ore, and scrap are charged in the quantities required to provide for oxidizing and fluxing the elements in the metallic portion of the charge, which may range from one-fourth scrap and three-fourths pig iron to two-thirds scrap and one-third iron.

The essential chemical reactions involved are expressed in their simplest form as follows:

Oxidation of silicon:

$$Si + O_2 \longrightarrow SiO_2$$

Oxidation of carbon:

$$C + O_2 \longrightarrow CO_2$$

Fluxing of silica:

$$SiO_2 + 2CaO \longrightarrow 2CaO.SiO_2$$

Oxidation and fluxing of phospoh

$$2P + 5O + 4CaO \longrightarrow 4CaO.P_2 \text{ ns}$$

Comb. of lime with sulphur:

$$CaO + S \longrightarrow CaS + O$$

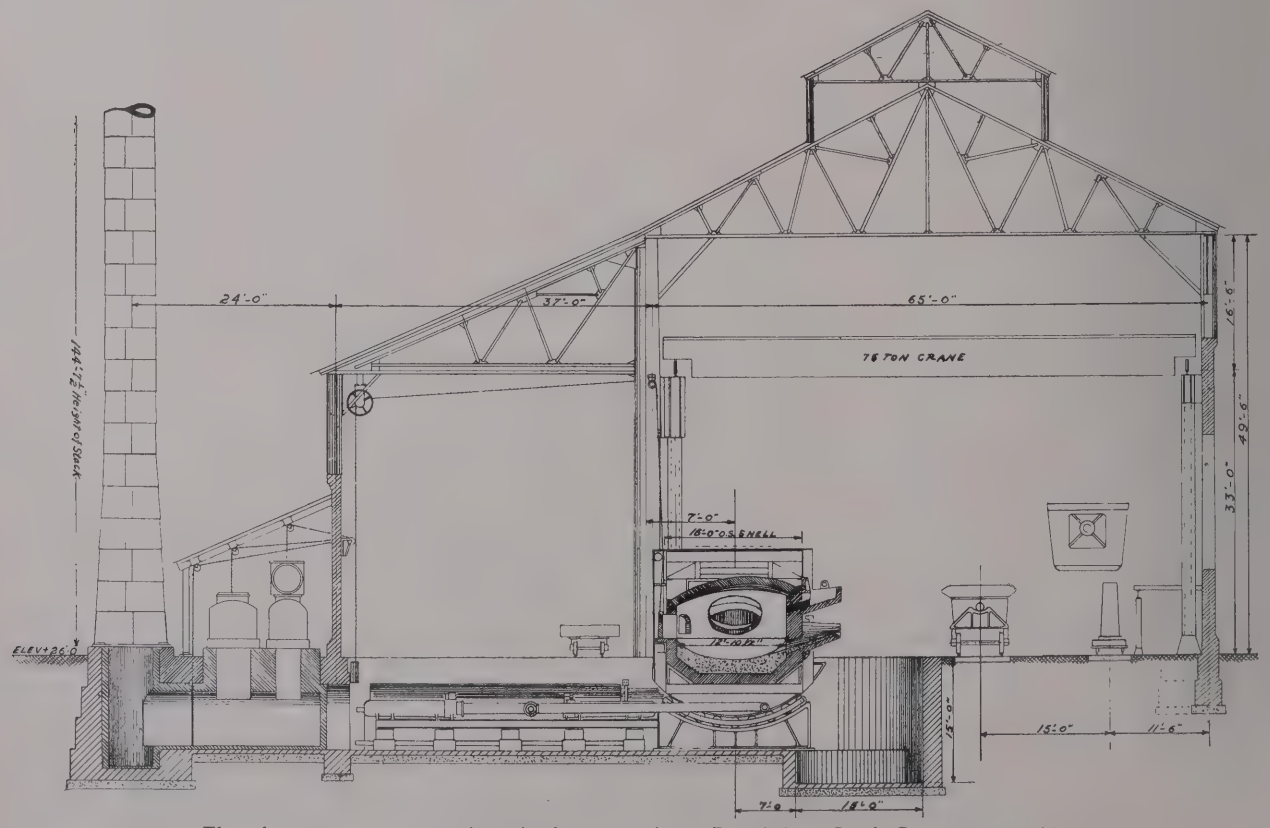


Figure 1.—50-ton open hearth furnace plant, Dominion Steel Company, 1901.

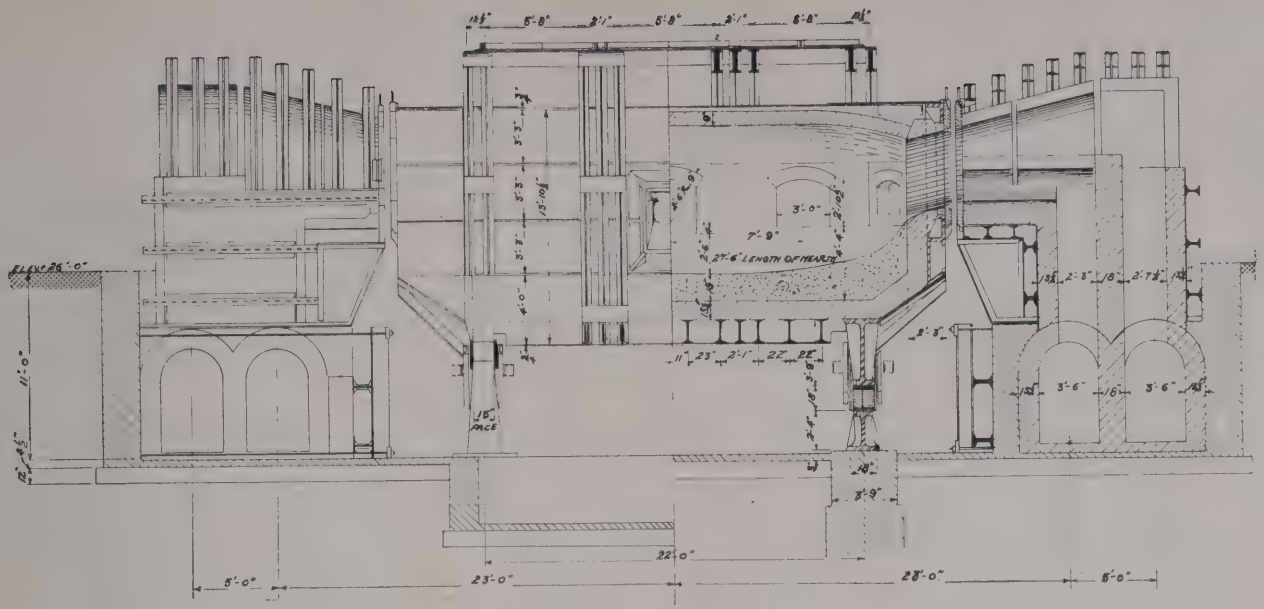


Figure 2.—50-ton Campbell tilting open hearth furnace, Dominion Iron & Steel Company, 1901.

At the commencement of the Sydney operations, the basic open hearth process was in its infancy and the mechanism of phosphorus removal and control of sulphur as outlined above was only imperfectly understood. In order to develop a suitable steel refining process it was thus necessary to solve these problems largely by trial and error.

The open hearth operators for the Sydney furnaces had been recruited largely from Pennsylvania, where the first basic steel had been made in 1888. Some had come from Trenton, Nova Scotia, where open hearths had been in operation using iron from the Ferrona blast furnace. Some Wabana ore had been used at Ferrona, but all operators faced entirely new metallurgical conditions in the new plant. The furnaces were of a new design, entirely different from the small stationary furnaces they were accustomed to work. The iron from the blast furnaces was so high in phosphorus as to be completely out of the range of their experience. In addition, as a consequence of blast furnace difficulties, they frequently had to work iron which was abnormally low in silicon and high in sulphur. This resulted in soft melts, with sulphur well outside steel specifications.

A further complication to open hearth operators was the difficulty encountered in the operation of the gas-producer plant with Cape Breton coals which were not well-suited for gas making. Interruptions in gas flow, and gas of poor heating value, were added handicaps to operations already rendered difficult by the raw materials available. Fuel difficulties led to experiments with coke-oven gas and by-product tar. In the latter case, the Dominion Steel Company was undoubtedly the first plant to burn liquid fuel in its steel furnaces. This development did not find common acceptance in the steel industry until twenty to thirty years later.

The problem of coal quality continued to be a serious one and rapidly worsened in 1903, when coal production from No. 6 mine began. This coal was found to contain well over 6 per cent sulphur and was very high in ash. The Steel Company refused to accept it because of the inferior coke it produced. The Coal Company maintained that, since it was from the Phalen seam, which had been designated by the Steel Company as the one from which their coal was to be supplied, no violation of contract could be claimed. On the refusal of the Steel Company to accept the No. 6 coal, the Coal Company cancelled the contract, forcing the Steel Company to secure coal elsewhere and to curtail operations.

A lawsuit ensued which was won in the lower courts in 1907 by the Steel Company. In appeals to the Supreme Court and later to the Privy Council, the decision of the lower court was upheld.

As a settlement of its claims, the Steel Company acquired control of the common stock of the Coal Company, a transaction which resulted in the formation of the Dominion Steel Corporation, a holding company for both industries, in 1910.

The Bessemer and Duplex Process at Sydney

In the first five years of operation, much progress had been made in solving the metallurgical problems which beset the industry. Ore and limestone were better prepared, while modification of blast furnace design and the introduction of furnace cooling gave smoother furnace operation, longer furnace campaigns, and a product of more uniform analysis. Coal washing had decreased ash and sulphur and improved the blast furnace coke.

A better understanding of the chemistry of the basic open hearth process brought the knowledge that the high silicon pig iron aggravated the ever troublesome problem of phosphorus removal in the open hearth furnaces.

The ease and speed with which silicon could be removed in the acid bessemer offered an attractive solution to the difficulty. It was believed that the acid bessemer converter could supply de-siliconized iron to the 50-ton open hearths and so speed up the output of these furnaces as to effect a substantial increase in production and a reduction in cost.

Accordingly plans were approved for an acid Bessemer plant, on which construction was begun in late 1906.

Plans called for three 12-ton converters with the necessary ancillary equipment. Only two were erected at that time. Bessemer operation began in May, 1907, and duplexing was begun immediately in the 50-ton furnaces.

In the Bessemer process, the charge is generally 100 per cent molten blast furnace iron through which air under pressure is blown. The reactions which take place are essentially those which occur in the open hearth process. The oxygen is supplied by the air of the blast

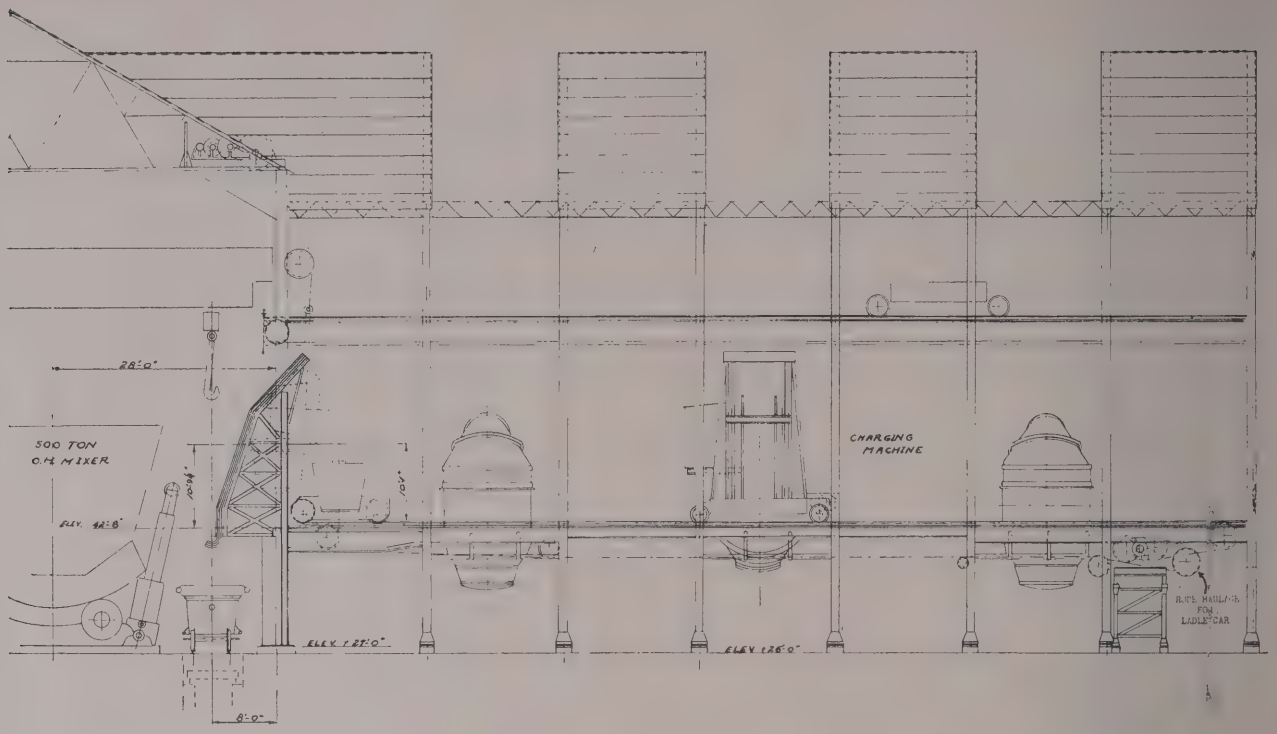


Figure 3.—Basic Bessemer plant, Dominion Iron & Steel Company, 1911.

and the reactions proceed with great speed.

In the acid Bessemer, so called because the vessel is lined with acid material, no lime is used and thus no removal of phosphorus or sulphur is possible, only silicon and carbon being eliminated.

The basic Bessemer, similar in construction and operation to the acid Bessemer, is however lined with dolomite, a basic material. The addition of burned lime to the vessel after the silicon and carbon have been blown out enables a stable compound of lime and phosphorus to be formed. Sulphur is also removed to some extent by the lime added to the slag. The process thus duplicates the basic open hearth in its essential reactions.

Operation of the acid Bessemer for the purpose intended was not the success that had been anticipated. Variations in pig iron analysis resulted in wide variations in the analysis and temperature of the blown metal. This gave rise to refining difficulties and delayed heats in the open hearth furnaces. The blast furnace metal was of such a composition as to give rise to undesirable operating conditions in the converters such as loss of metal by spitting, and the building up of troublesome concretions on the converter lining.

After a period of some fifteen months of unsatisfactory operation, acid Bessemer blowing was suspended.

In the meantime, plans had been made to change the plant over to basic Bessemer operation. Basic Bessemer blown metal for duplexing was first produced on Sept. 26th, 1908, and continued to November 13th of that year on an experimental basis. Acid practice was resumed on November 18th and continued until March 7th, 1909.

When the additional equipment for the basic Bessemer operation had been installed, production began in April, 1909.

The blown metal, approaching the analysis of mild steel, was charged into an open hearth furnace. Since most of the metallurgical work, *viz.*, the removal of silicon, carbon, and phosphorus, had been effected in the Bessemer converter, little more remained to be done than to accumulate a full charge in the 50-ton furnace, make the necessary alloy additions, adjust the temperature, and tap the heat.

A considerable tonnage of steel was made by this process, but, for many purposes, the quality of the steel was found to be inferior. Moreover, since the plant had originally been designed for acid Bessemer operation, the installation of the additional equipment required for basic operation resulted in a cramped and inefficient plant. High scrap losses in transferring the molten metal to the open hearth furnaces reduced the percentage yield of duplex ingots from the blast furnace iron to uneconomically low figures.

The basic Bessemer had the advantage of producing, as a by-product, the so-called Thomas slag, high in soluble calcium phosphate. This was processed by the Cross Fertilizer Company, of Scotland, which had established a plant at Sydney in 1911. Unfortunately, as a result of the high silicon in blast furnace iron and the difficulty of effecting proper separation of the blown metal and slag, the composition of the latter for use as fertilizer was inferior.

Further modifications of Bessemer practice in conjunction with final refining in the open hearth furnaces were attempted with moderate success.

One such practice, which was equivalent to using the basic converter for producing acid blown metal, was operated from December 1st, 1909, to July 31st, 1910, with no outstanding results. Basic blowing was continued from that date to January 31st, 1911.

Active Mixer Operation

The difficulties arising from fluctuating basic iron analysis, to which basic Bessemer practice was subject, had been overcome in certain European districts, notably the northeast coast district in England, by the installation of large mixer furnaces in which some refining could be carried out.

Operating officials of the Dominion Steel Corporation visited England in 1908 and 1909, and brought

from the North Eastern Steelworks, Middlesborough, plans for the construction of two furnaces of this type, each of 500 tons capacity. These furnaces were, in general, similar in design to the 50-ton open hearths though of ten times the capacity. They tilted in the same manner and were fired through regenerators with producer gas. In these furnaces by the use of iron oxide for oxidation and lime for fluxing, it was hoped to reduce the varying silicon content of the blast furnace iron to a uniform low level. This would shorten the basic Bessemer blowing time as well as improve the quality of the phosphoric slag for fertilizer. In addition, it was planned that these furnaces would be able to de-siliconize all metal for steel making, and thus accelerate the open hearth process by reducing the amount of oxides and fluxes required.

The first of two furnaces, *A* and *B*, went into operation on January 19th, 1913, and the second in May of the same year. Their performance is indicated in Table II.

TABLE II

	Si	S
Average metal charged	1.20%	0.050%
Average metal tapped..	0.76%	0.045%
Percentage removed.	37%	25%

COMPOSITION OF SLAG PRODUCED:

SiO_2.....	37.00%
CaO.....	41.50%
P_2O_5.....	2.30%

MATERIALS USED FOR REFINING:

Lime.....	1.33%	of total iron
Iron Oxide...	1.37%	charged

OUTPUT: 450 gross tons per furnace per day

The experience gained in this operation showed that:

(*a*) Silicon was not reduced to the level expected.

(*b*) Erosion on furnace linings was excessive because of the silicious slag produced.

(*c*) The expected improvement in Bessemer slag failed to materialize because of the difficulty of keeping the silicious slag from the mixer furnaces out of the Bessemer converters.

(*d*) Scrap losses due to multiple handling of molten metal were high and ingot yields were correspondingly low.

(*e*) This process produced steel suitable for soft steel only. Rail steel made in this way was unsatisfactory.

BERTRAND-THIEL PROCESS

After the first trials on de-siliconizing in the active mixers failed to prove practical, a modified practice was instituted in October, 1913. This practice known as the Bertrand-Thiel process had been developed in Bohemia for handling high-phosphorus pig iron and appeared to be suited for the refining of Sydney basic iron. One mixer furnace was operated as a de-siliconizer and supplied metal to the second furnace to be de-phosphorized. The results produced are shown in Table III.

The Bertrand-Thiel practice failed to produce sufficient reduction in phosphorus to warrant the cost of operating two furnaces. Furthermore, it was no help in reducing sulphur. Furnace production was low, and the practice was soon discontinued.

After this period, one of the furnaces was put on Talbot practice for a short time. This process will be described in detail later in the paper.

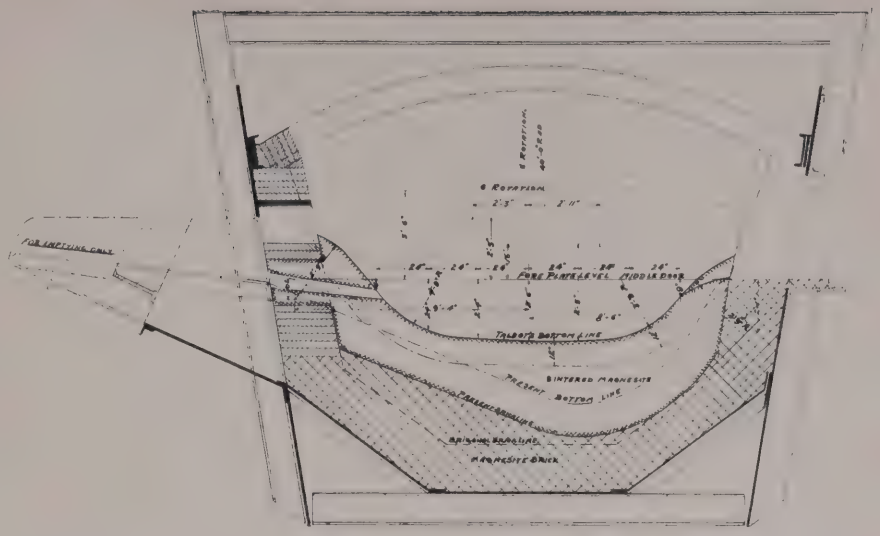


Figure 4.—Active mixer furnace, showing changes in bottom lines for different practices, Dominion Steel Company, 1912-14.

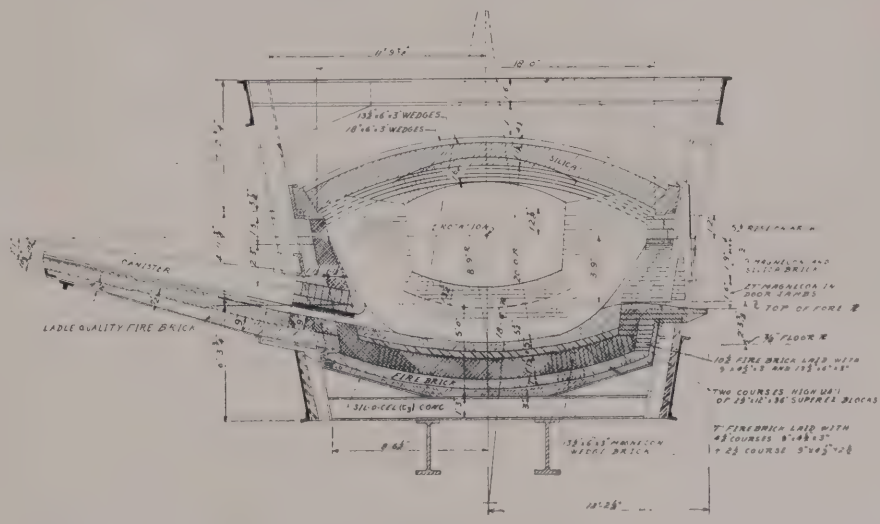


Figure 5.—Cross-section through 100-ton tilting open hearth furnace, Dominion Steel Corporation, 1939.

In March, 1914, the 500-ton mixer furnaces were equipped with closed spouts, designed to deliver the metal through a nozzle and stopper arrangement, and so prevent the slag from the de-siliconizing furnaces reaching the Bessemer converters.

It was found possible to reduce the silicon at this time to about 0.60 per cent and since no silicious slag was reaching the converters, slags with 20 per cent P_2O_5 could be produced compared with 17 per cent maximum when no effective separation was made. The stoppered spouts could not be maintained in working condition, and, moreover, higher silica slags gave rise to dangerous erosion of the mixer furnace lining. The practice was therefore abandoned after a few weeks' trial.

A further modification of practice was put into operation in April, 1914, when one furnace was put on to de-phosphorize metal which had

been blown in the basic-lined Bessemer converter to the point where silicon only was eliminated. The second furnace adopted this practice a month later. Six of the 50-ton furnaces were used to finish the metal from the large furnaces. The results shown in Table IV were obtained.

In the 50-ton furnaces, 30,000 lb. steel scrap with 15,000 to 17,000 lb. limestone was charged and 95,000 to 100,000 lb. de-phosphorized iron added.

No spectacular outputs from the 50-ton furnaces were achieved on this practice. The low yields which inevitably accompany repeated handling of molten metal were a serious handicap. Rolling difficulties encountered in the mills were an added problem, and caused an early cessation of this development.

With this practice, every possible combination of Bessemer, active mixer, and open hearth had been exhausted without any outstanding success, and it was decided to shut down the Bessemer plant and convert furnaces *A* and *B* to 100-ton open hearth furnaces.

Operation of Furnaces *A* and *B* as Open Hearths

The shop in which the active mixer furnaces were located was designed for this practice only and thus was by no means suited for regular ingot production.

The wall of the tapping side of the building was so near the furnaces as barely to allow room for one standard-gauge track on ground level below the furnace tapping spout. It was necessary to tap the furnace charge of 100 tons successively into two 50-ton ladles placed on railway trucks, and to transport these into the 50-ton furnace shop for teeming into ingots.

The difficulty of controlling the composition of the second half of the heat, tapped after an interval of half an hour, was of constant concern. However, after some experience had been gained, the practice was followed with good success through 1939, when a pouring bay was constructed and facilities provided for operating the shop as a self-contained unit.

It is remarkable that in the twenty-five years of operation of these converted furnaces, during which several thousand ladles of steel were moved, there was not a single derailment involving a ladle of molten steel. Such an occurrence would have been a major problem, since no railway crane was available to lift such a load.

Shortly after the declaration of war in August, 1914, all plant operations were shut down.

When steel making was resumed in September, 1914, the 50-ton furnaces began making steel from all-cold charges, using one-third scrap and two-thirds cold pig with 25 per cent limestone and 2.5 per cent ore. During this period, experimental heats were made using about 4 per cent of manganese ore in the charge. This work continued when the blast furnaces began producing iron toward the end of the year. It was found that the high residual manganese in the melt caused difficulty in the removal of phosphorus, but a considerable saving was made in the amount of ferromanganese required. Rolling and inspection results were good, but all these advantages were more than offset by the high cost of the manganese ore required, and the practice was dropped.

A comparison of regular basic practice on the 50-ton furnace in 1906 and 1914 is given in Table V.

Consumption of limestone per unit of blast furnace iron was increased, while oxide requirements were unchanged. However, longer refining times had reduced furnace outputs, although the practice had resulted in material improvement in steel quality and better control of phosphorus and sulphur.

The Hoesch Process

Early in 1915 a modified duplex process, which had been developed at the Hoesch works in Germany, was introduced into the 50-ton furnace shop. This practice, really a modification of the Bertrand-Thiel process, used one furnace to carry out steel refining in two stages.

Limestone, amounting to about 12 per cent of the metallic charge, and iron oxide to about 12 per cent, were charged on the furnace hearth, and followed as soon as possible by a charge of 50 tons of hot metal. No further additions were made and metal and slag were tapped out in about four hours, and held in the ladle.

The empty furnace was then charged with 16 per cent lime or limestone and 5 per cent iron ore. The metal was then returned to the furnace and refining proceeded with. Usually, additional ore up to 8 per cent was required and 500 to 800 lb. of fluorspar was used to thin the slag. The heat was tapped about ten hours later.

Table III.— Results with Bertrand-Thiel Process

	Si	P	S
Metal charged to 1st furnace	1.20%	1.50%	0.065%
Metal delivered from 1st furnace	0.85%	1.55%	—
Metal delivered from 2nd furnace	0.10%	0.70%	0.060%

Composition of Slag from 2nd Furnace:

Silica	20.00 per cent
Lime	32.00 "
Iron oxide	9.00 "
Phosphoric acid	15.00 "

Total Materials Used in Both Furnaces:

Lime	6.6% of metal
Oxides	13.8% of metal

Output from 2nd Furnace: 192 tons per day

Table IV

	C	Si	P	S
Blown metal charged	3.00%	0.10%	1.60%	0.050%
Metal delivered	2.50%	0.05%	0.65%	0.070%

Composition of Slag Produced:

SiO_2	21.44%
CaO	37.09%
P_2O_5	15.44%

Metal Throughput: 325 tons per furnace per day

Table V.— Comparison, 1906 and 1914 Practice

	1906	1914
Scrap charged	24,790 lb.	54,280 lb.
Cold pig charged	740 lb.	3,770 lb.
Hot metal charged	88,140 lb.	68,560 lb.
Total oxides	22,800 lb.	18,300 lb.
Total limestone	29,580 lb.	27,800 lb.
Lbs. oxide per ton metal	510 lb.	510 lb.
Lbs. limestone per ton metal	665 lb.	770 lb.
Aver. phosphorus.	0.030	0.024
Aver. sulphur	0.044	0.034
Average tons/ furnace/day	92	83.5

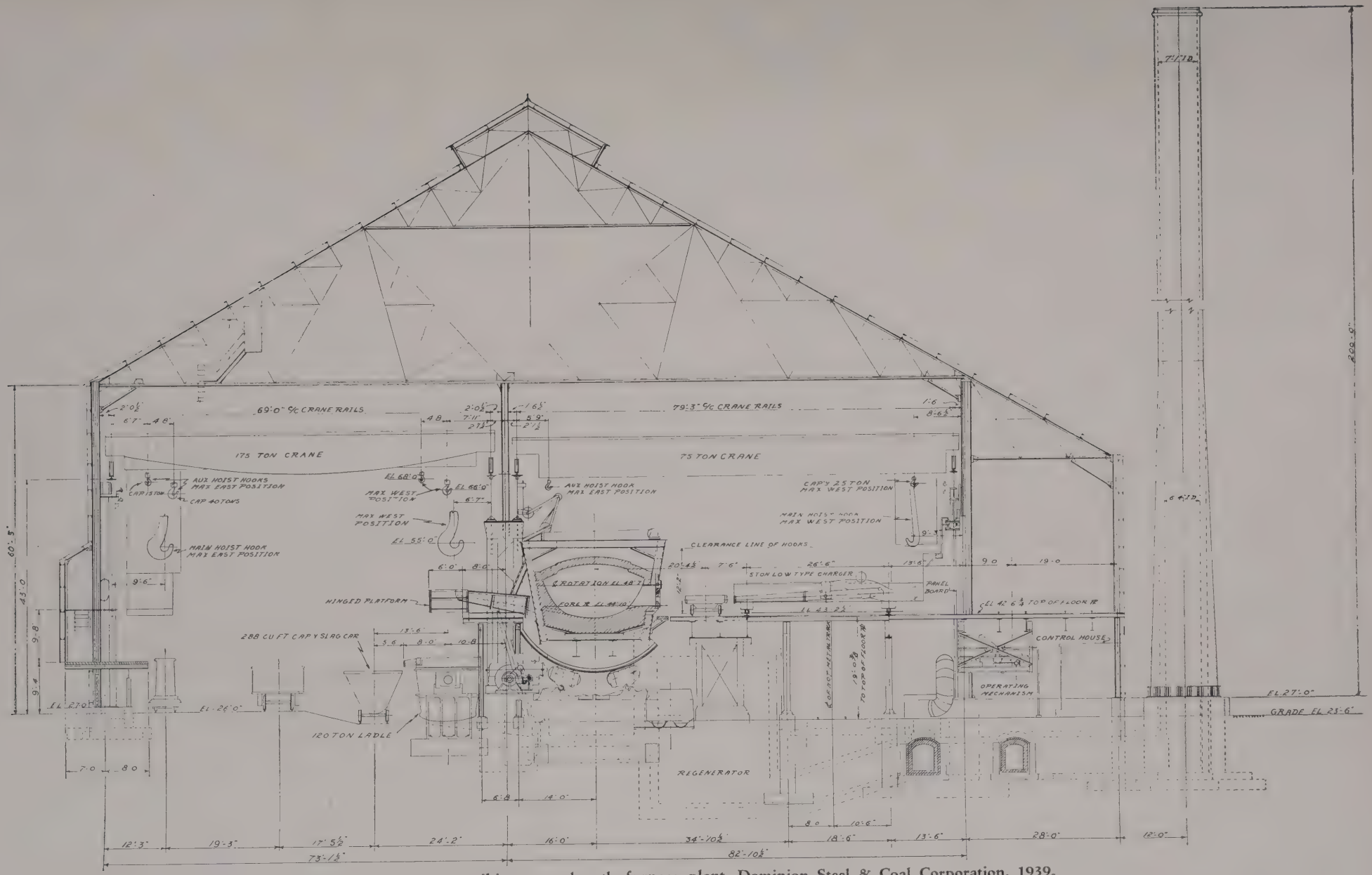


Figure 6.—100-ton tilting open hearth furnace plant, Dominion Steel & Coal Corporation, 1939.

The aim of the Hoesch process was to effect removal of the silicon and part of the phosphorus of the charge in the first stage, while retaining most of the carbon. By tapping after the first period, the metal could be separated from the siliceous slag and returned to the furnace for the removal of phosphorus under a fluid, oxidizing, highly basic slag, and for adjustment of composition and temperature.

The slag produced in the second period was of no value as a fertilizer because of the necessity of using fluorspar to obtain slag fluidity.

The Hoesch process operated with little or no scrap in the charge and thus required a plentiful supply of molten iron, which was not always available. Furthermore, the multiple handling of metal contributed to excessively low yields and increased handling cost. The consumption of limestone and oxides was very high and heating costs and heat times were well above normal practice. These shortcomings rendered the process uneconomical.

Bessemer operations were resumed in May of 1915 and continued supplying basic blown metal to one or two of the 50-ton furnaces for duplexing. Other furnaces continued to operate on straight basic practice.

All operations were seriously threatened by the lack of basic materials — magnesite brick and grain magnesite for furnace maintenance. Normally, these materials were manufactured from Austrian magnesite, but when this became unobtainable during the war, substitute materials were made from domestic magnesitic-dolomite, and the industry continued to operate under this handicap until the end of the war. It is of interest to note that a Canadian magnesite industry had its beginning in this trying period.

The Talbot Process

In 1900, Henry Talbot published the details of a practice he had developed for the production of steel from high-phosphorus pig iron, using a tilting furnace. When the heat is ready to tap, a portion only of the steel is taken out of the furnace. To the steel remaining, lime, ore, and scrap is added, and a new supply of molten is poured in.

This practice was begun in 1920 in Sydney under the direction of Mr. C. H. F. Bagley, who had come over from England to supervise it.

The 100-ton furnaces were put on this process, using the following standard charge:

Burned lime	20,000 lb.
Iron ore	30,000 lb.
Scrap	30 to 33 tons
Hot metal	70 to 75 tons

Additional lime, averaging 10,000 lb., and about 15.000 lb. of ore, were worked into the heat for finishing, and fluorspar was used to thin the slag.

Output of Talbot steel on the 100-ton furnaces averaged 163 tons per day, considerably better than the 134 tons produced on straight basic open hearth practice. It was found, however, that the quality of the steel was not all that could be desired. Sulphur was frequently a problem and some 30 per cent of the heats finished outside of the sulphur specification. The process was a heavy consumer of blast furnace iron, but yields were higher than those of the duplex processes hitherto attempted, because of the heavy consumption of iron ores. With ore available at low cost, the process had attractive features for the production of low-carbon steels. It could not, however, be used for the production of the rail steel which made up the bulk of the steel output.

The use of the Talbot process in conjunction with blown metal from acid-lined converters was considered to have interesting possibilities, particularly in the direction of making high-phosphoric-acid slag for fertilizer. Plans were therefore made in 1921 to rehabilitate the Bessemer plant for acid blowing and to redesign furnaces *A* and *B* under the direction of Mr. Talbot. Curtailment of operations later in that year caused this plan to be shelved and it was never revived.

Further Development of the Open Hearth Process

Through the years, the matter of supplying a phosphoric slag with a high content of soluble phosphate had been constantly before the operating heads of the Sydney plant. Indeed, it had influenced to a considerable extent the choice of steelmaking practices. The slag produced in the basic open hearth process as then carried out was considered unsuited for fertilizer because it was low in phosphoric acid and of very low solubility.

A process was in use at the Frodingham Works, in Scunthorpe, Lincolnshire, England, by which a soluble high-phosphoric slag could be produced in the open hearth. The practice involved the use of an active mixer to reduce the silicon in the blast furnace metal to 0.10-0.20 per cent. In the open hearth furnaces, the tapping slag from the previous heat was held in the furnace with some 20,000 lb. of metal. Through this slag, rich in lime and oxides of iron, and at a high temperature, the de-siliconized metal was poured. The result was the rapid formation of a basic slag containing about 18 per cent phosphoric acid, almost all of which was soluble, since no fluorspar was used.

The process was attempted at Sydney with raw molten iron, but was not successful. It was found that the high-lime slags necessary to get low phosphorus and sulphur in the finished steel could not be rendered sufficiently fluid without the use of fluorspar. This, of course, defeated the principal purpose for which the process had been developed.

Flush Practice

There were strong indications, however, that the flushing of slag immediately after the metal addition resulted in more rapid melting and greater ease of removal of phosphorus in the finishing period of the heat. Some elimination of sulphur was also noted. Work was therefore continued with particular emphasis on the early flushing of slag. Attempts to hold the tapping slag were soon discontinued, and the furnaces were completely drained at each tap.

As the process evolved, it incorporated many of the elements of the Hoesch practice tried years before. The flush period corresponded to the first stage of Hoesch heats, removing nearly all the silicon and about 25 per cent of the phosphorus of the charge. The average composition of the flush slag was somewhat similar to that of the first Hoesch slag, being as follows:

Silica	25—30%
Lime	30—40%
Phosphoric acid	11.00%

After the flush, the slag was enriched by lime additions, and kept fluid by the use of fluorspar, sufficient ore having been charged to oxidize most of the phosphorus and carbon. During this stage, the limestone charged rapidly decomposed and caused a vigorous boil by the evolution of its carbon dioxide. The result was the formation of a well dissolved slag high in lime, and sufficiently oxidizing to hold the phosphorus in combination. Since it was maintained high in lime throughout

the period after flush, it gave good control of sulphur.

As experience was gained in the proper control of this flush practice it became evident that a considerable step forward had been taken, and that this method of steel making appeared to be well suited to the processing of Sydney basic iron. It was found to be of the greatest importance to remove as great a volume as possible of the first slag, and charging of the furnace was carried out with this in mind. A notched door sill for slagging off was installed in the centre door of the furnace and provisions were made for the proper handling of the flush slag.

A typical heat was made as follows:

Start to charge	8.00 p.m.
Finished charge	8.20 p.m.
Hot metal addition	10.30 p.m.
Slag flush	10.50/11.30 p.m.
Lime addition	1.40 a.m., 1,500 lb.
Fluorspar addition	3.35/3.40 a.m., 600 lb.
Ore addition	4.10/4.35 a.m., 4,000 lb.

Limestone	14,500 lb.
Swedish ore	20,500 lb.
Manganese ore	2,000 lb.
Scrap	28,900 lb.
Metal	89,000 lb.

Metal analysis	Si = 1.45%	S = 0.060%
Analysis of metal at tap	P = 0.035%	S = 0.054%
Time of heat	9 hr., 30 min.	

Tapping slags analysed about as follows:

Silica	10—12%
Lime	53—55%
Phosphoric Acid	10—12½%

The success of this practice is indicated by the rapid jump in furnace output. In March, 1927, with this practice, the 50-ton furnaces averaged over 100 tons of steel ingots per operating day, a figure never before attained.

With some modifications which did not alter its character, flush practice continued as the standard steel making process at the Sydney steel works. While unimportant tonnages of Talbot steel were made in furnaces *A* and *B* when conditions warranted, these furnaces, too, operated successfully on the new practice, and produced tonnages which exceeded their best previous performances.

The Present Status of Flush Practice

Flush practice has proved flexible enough to be equally well suited for the production of high-quality steel of all grades from dead-soft to high-carbon. The flush slag, which removes from the furnace most of the silica formed by the oxidation of the silicon in the iron charged, is the balance-wheel that smooths out variations in analysis of blast furnace iron. When silicon is above normal, sufficient oxides are charged for its oxidation and a large volume of silicious slag is drained from the furnace. As the silica is removed, lime enters the slag and automatically ends the flush period by increasing slag viscosity. With low-silicon iron, which is frequently accompanied by high sulphur, the volume of the flush is considerably reduced. For this type of iron, the ore charge is cut down. The resulting slag thus remains high in lime throughout the melting and refining of the heat and thus contributes to reduction of sulphur.

The Firing of Fuel Oil

In March, 1947, the first 100-ton furnace was converted to the firing of *Bunker C* fuel oil, and by the end of 1950 all five furnaces in No. 2 shop and five of the original 50-ton furnaces in No. 1 shop were on liquid fuel.

Flush practice has been even more effective under the changed conditions. The higher furnace temperatures possible with fuel oil accelerated the furnace reactions, increased the effectiveness of the flush, and facilitated the formation of the final slag. These factors, together with the use of low-sulphur fuel oil, have made it possible to reduce the lime required and thus decrease slag volume and increase the effective melting capacity of the furnaces. The elimination of producer gas, with its high sulphur content, has resulted in much lower sulphur in finished steel.

Finally, it has been found possible to increase the melting capacity of the old active mixer furnaces *A* and *B* to about 165 tons, and these furnaces now produce daily tonnages of close to 300 tons of ingots per furnace. The Sydney open hearth shops are now producing, from 10 open hearth furnaces, 20 per cent more tonnage than formerly made in 15 furnaces.

Conclusion

Flush practice has been universally adopted by the steel industry, and in North America has become in recent years the standard process for the production of basic open hearth steel from charges containing high percentages of hot metal. The many stationary open hearth shops which formerly operated on high scrap charges have been driven by the dwindling supplies of steel scrap to higher and higher percentages of hot metal, and as a consequence to the double slag process. Although the stationary furnace is not the ideal medium for its most effective use, flush practice has enabled these plants not only to maintain but to materially increase ingot outputs per furnace.

This practice, as now carried out in the Sydney open hearth furnaces, is the evolution of fifty years of trial with every known method of making basic steel. It has been developed into the most flexible and efficient means yet devised of solving what is undoubtedly one of the most difficult problems in the whole industry.

Acknowledgments

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Significant Developments in the Canadian Oil and Gas Industry During 1951

By W. B. DINGLE *

(*Ottawa General Meeting, January,* 1952)

(*Transactions, Volume LV,* 1952, pp. 216-222)

Introduction

MANY PAPERS presented on developments in the Canadian Petroleum Industry during the past few years have commenced with this general sentence: "The past year has been a record one for the oil industry in Canada, with new records being set in nearly all phases of exploration and development activity". This sentence applies equally well to 1951. It is not possible to outline all the details of the broken records in exploration, development, and production but general comparisons can be made. There are several significant developments that have stolen the spotlight during 1951, some of which were foreseen in 1950 and some which were not. Among the highlights were the following:

(1) The greatly expanded activity in the search for gas, which was touched off by the new Alberta regulations covering gas leases and reservations.

(2) The discovery of commercial oil production in the Williston basin of North Dakota and Manitoba, which will have far reaching effects on Canadian oil.

(3) The greatly increased activity in northern Alberta, northwestern British Columbia, the southern Northwest Territories, Saskatchewan, and Manitoba.

(4) The new discoveries in the general central Alberta area.

(5) The granting of a permit for an oil pipeline to the Pacific coast.

(6) The arrival of Western Canadian crude in central Ontario.

(7) The number of new refineries completed, commenced, or proposed.

(8) The present and proposed expansion in the petrochemical industry.

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Most of the remarks made on various exploration and development activities will refer to Western Canada operations, primarily because the greatest activity and the most significant developments have been there. Eastern Canada exploration and development operations were at a slightly higher level than in 1950.

Exploration Activity

Exploration activity has shown much the same rate of increase as it had in years prior to 1951 (Figure 1). Geophysical work and the number of wildcat wells have shown large increases, with geophysical activity increasing 25 per cent from 1,300 crew-months in 1950 to 1,630 crew-months in 1951, and the number of wildcat wells by 51 per cent from 384 in 1950 to 578 last year. Geological exploration increased 64 per cent between 1950 and 1951 and, if compared with the figures for the past five years, this type of exploration shows very significant gains. Analyzing the geological exploration further, it is noted that surface geological work diminished but that core drilling shows a 181 per cent increase. This probably indicates that the exploration people are now endeavouring to supplement their geophysical work with shallow structural drilling, to obtain a better subsurface picture, before commencing any deep drilling programme.

As in previous years, Western Canada, and particularly the Province of Alberta, continued as the second most active area in the world with regard to geophysical activity, surpassed only by the State of Texas. During the year, a peak of 153 parties in the field was reached, with an average of 134.

Total exploration expenditures reached an all-time high, with 135 million dollars being poured into geophysical and geological exploration, wildcat drilling, reservation acquisition, and related work (Figure 2). More and more of the exploration dollars were being spent in the northwestern part of central Canada, and small oil discoveries at Belloy in the Peace River area of Alberta and near Fort St. John in British Columbia have accentuated this trend.

The greatest exploration activity in Canada has been in what is locally referred to as the 'Fairway', that general area extending from the vicinity of Edmonton through Joseph Lake, Armena, Camrose, Stettler, Big Valley, and as far south as Drumheller. Significant discoveries have been recorded all through this region.

The discovery of oil in North Dakota and northeastern Montana in the Williston basin has caused considerable shifting of exploration activity. These discoveries, together with that in Manitoba, are in the Devonian and Mississippian and, as a result of them, the area favourable for oil production has been greatly enlarged. They have encouraged intensive exploration activity in southern Saskatchewan and southwestern Manitoba. Geophysical crews have doubled both in Manitoba and Saskatchewan, going from 588 to 1,194 crew-months in Saskatchewan in the six-month period before and after the North Dakota discovery and from 120 to 268 crew-months in Manitoba. Acreage taken up has increased by 7 million acres in Saskatchewan and by 2 million acres in Manitoba.

Overall acreage acquisition in Western Canada increased by 36,000,000 acres, from 130,000,000 acres to 166,000,000 acres. This increase is the same as that recorded in 1950. Significant is the fact that 4,000,000 acres were acquired in the Northwest Territories and 10,000,000 acres in British Columbia. (The figures presented here are ap-

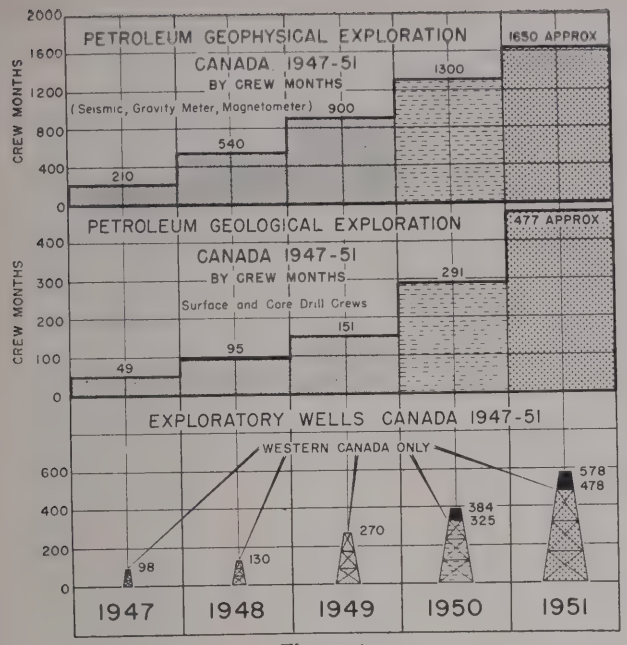

Figure 1.

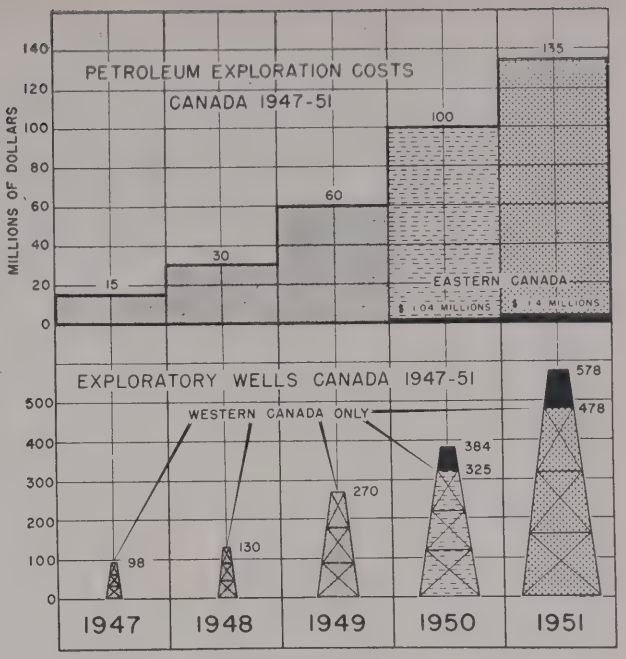

Figure 2.

proximate only as data of this type are difficult to check).

Development Activity

General development activity has continued at a peak slightly ahead of 1950 (Figure 3). Total development costs, however, were below those of 1950. In the main, this is explained by the decrease in monies paid for proven reserves; for example, $32,000,000 was paid to the Alberta Government for Redwater Crown reserves in 1950 whereas only $3,000,000 was paid in 1951. Another factor accounting for some of the decrease was the much lesser amount of development drilling at Redwater in 1951, wells completed in that year numbering only 176 as compared with 448 in 1950. This decrease in Redwater development was offset by greatly increased development elsewhere.

With this completion, for all practical purposes, of drilling, Redwater became the first of the major fields to be discovered in Canada in recent years to pass from a predominantly developing to an operating field. Highlighting this change are the problems of salt-water disposal, and possibly of pressure maintenance, which have arisen. The fact that the producers at Redwater have already taken measures to meet and solve these problems is reassuring evidence that the Petroleum Industry is accepting its responsibility for the efficient production of our petroleum resources.

Development of the Leduc field continued at an expanded level, with up to forty rigs being employed in drilling at various times during 1951. This activity brought about numerous extensions of the field proper and discovery of local productive areas to the southeast, south, and southwest. General Leduc activity was complemented by

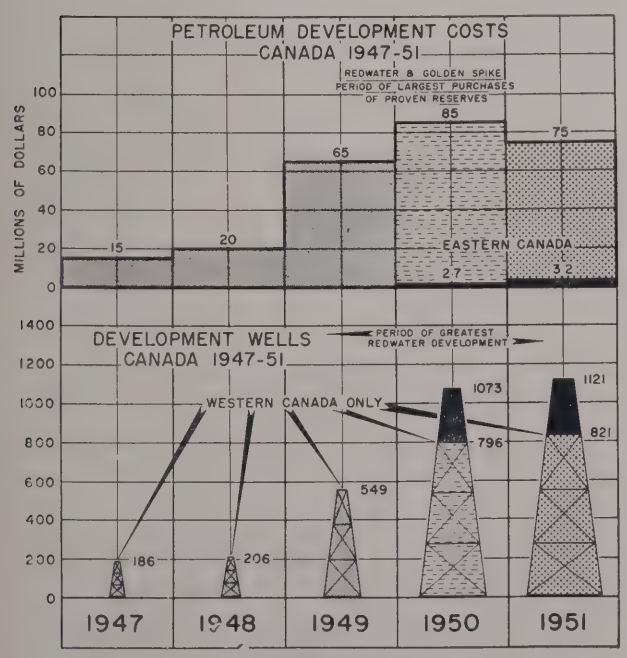

Figure 3.

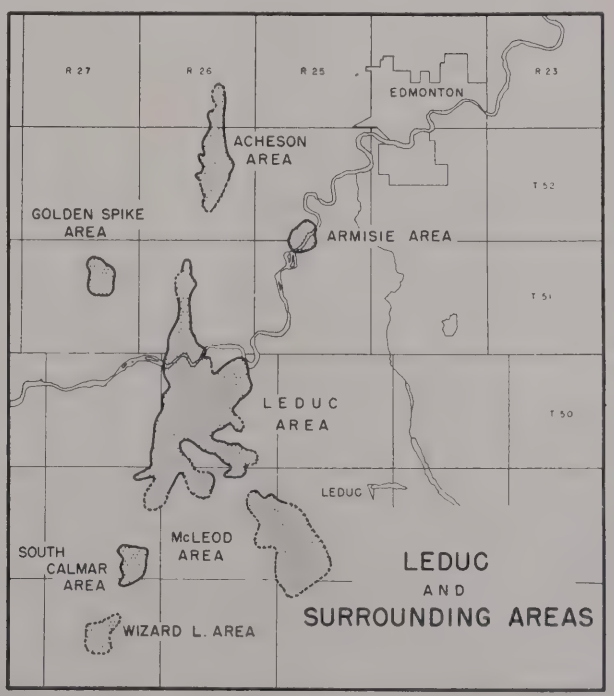

Figure 4.

discovery or development of fields surrounding it, namely, Wizard Lake, South Calmar, Armisie, Acheson, Stony Plain, and Golden Spike (Figure 4). In the general area, there are now some 878 producing wells.

Considerable development of existing fields took place at Joseph Lake, Stettler, and Big Valley. Acheson developed into a much larger field than was foreseen in late 1950 and present indications are for possibly 130 wells in this field.

Development in most fields was slowed somewhat during 1951 by the lack of drilling rigs. While up to 220 rigs have been employed in Western Canada, the demand has been in excess of this, especially for rigs for exploratory work, and consequently some development work has been shelved.

With the enactment of new Alberta regulations covering natural gas leases and reservations, and in order to ensure reserves sufficient to justify future gas export and adequate to meet the anticipated increased demand for gas for petrochemical industries, a great deal of gas exploratory and development work was carried on in 1951. Some companies were engaged solely in such work, which resulted in the finding of substantially increased gas reserves, notably at Morinville, Legal, and the general Cessford area. With the completion of the gasoline plant at Jumping Pound and the transportation of gas from this field to markets in Calgary, Exshaw, and Banff, further development of this previously dormant gas field can be expected.

Developments in Eastern Canada continued on a slightly expanded scale over 1951, and, while development expenditures there are low compared to Western Canada, it is significant to note that some 300 wells, one of which is in New Brunswick, were drilled. Most of these are gas wells. In the middle of 1951, some interest was shown in a reported oil discovery in Lambton county, Ontario.

TABLE I.— INDICATED SUCCESSFUL OIL AND GAS EXPLORATORY WELLS, 1951

PROVINCE	OIL DISCOVERIES			GAS DISCOVERIES		
	New fields	New pools	Extensions	New fields	New pools	Extensions
B.C.	1	—	—	—	—	2
Alberta	21	6	20	31	4	27
Saskatchewan	2	1	2	2	1	—
Manitoba	1	—	2	—	—	—
N.W.T.	—	—	—	1	—	—
TOTALS	25	7	24	34	5	29

NEW DISCOVERIES

Discoveries during 1951 have not, at least from present indications, uncovered any more 'Redwater' fields, but they have been numerous and scattered and include both gas and oil and range from Manitoba to northeastern British Columbia. During the year, 56 oil and 68 gas discoveries were made (see Table I). These include new pools as well as extensions to known pools. The 56 indicated oil discoveries are broken down to 25 new fields, 7 new pool discoveries, 24 extensions. The 68 gas discoveries include 34 new fields, 5 new pools, and 29 extensions. In Table II, these new discoveries are indicated by geological zone of discovery.

Some of the more promising oil discoveries are at Camrose, Armena, Bashaw, New Norway, Clive, Caprona, and Drumheller in the general 'fairway' area; Wizard Lake, South Calmar, and Armisie in the general Leduc area; in the region immediately north of Edmonton; at Belloy and Fort St. John in the Peace River area; at Eatonia and Coleville in Saskatchewan; and at Virden in Manitoba (see Figure 5).

TABLE II.— INDICATED SUCCESSFUL OIL AND GAS EXPLORATORY WELLS, WESTERN CANADA, 1951, BY GEOLOGICAL ZONES

ZONE	OIL	GAS
COLORADO:		
Viking	4	14
Other	1	12
BLAIRMORE	18	37
JURASSIC	—	1
TRIASSIC	—	2
PERMO-PENN.	2	—
MISSISSIPPIAN	6	3
DEVONIAN:		
Nisku	16	4
Leduc	7	1
Cooking Lake	2	—
Mis-Devonian	—	1
Unclassified	—	2
TOTALS	56	77*

*Includes one 4-zone, one 3-zone, and four dual-zone wells.

Of the above mentioned discoveries, several warrant particular mention. The indicated oil pay zone at Wizard Lake is the thickest yet recorded in Canada, some 612 feet in

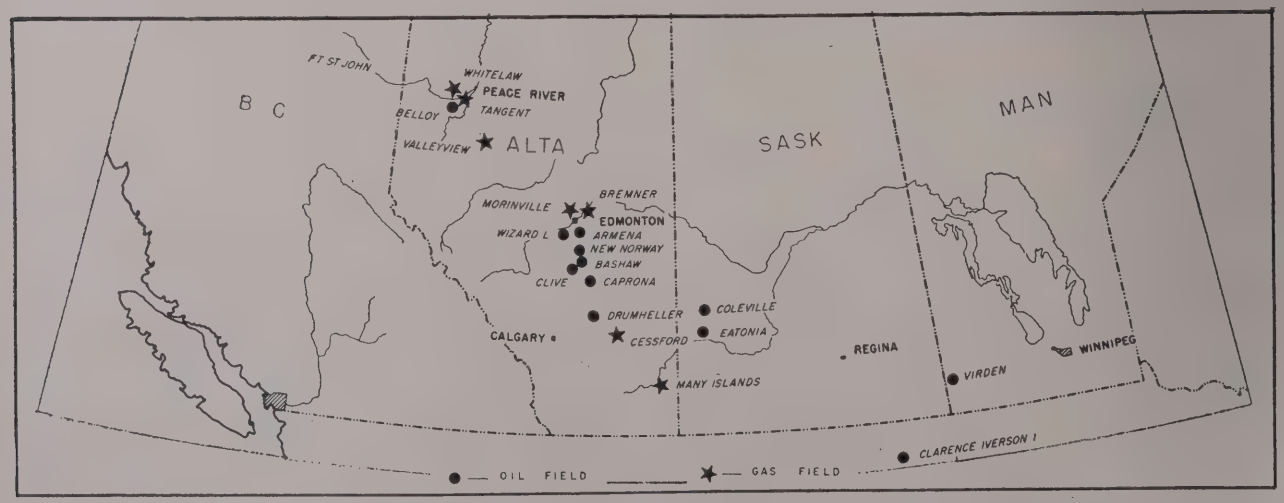

Figure 5.—Map of western Canada, showing location of some major developments in 1951.

the *D-3* zone, which is slightly more than that recroded at Golden Spike in 1949. While it is yet early to give reliable figures, certainly this field should greatly exceed 50,000,000 barrels of oil. Lower Cretaceous and *D-2* production is also indicated.

At South Calmar, a pay thickness of over 400 feet has been recorded, also in the *D-3*. This area is only three miles northeast of Wizard Lake. It is not believed that these fields are joined.

The discovery at Armena is from the Viking sand — the same producing zone as in the Joseph Lake field. While it is too early in the development to attempt a reliable estimate of the size of the field, it at least gives indications of being equal to or larger than Joseph Lake; certainly, the average pay thickness appears to be greater. The discoveries at New Norway, in both *D-2* and *D-3*, and at Caprona in the *D-2*, are particularly important as they gave fairly thick pay zones, 60 to 100 feet, and at New Norway the Leduc, or *D-3*, also was productive from a zone of some 60 feet.

With Devonian production at Duhamel, New Norway, Clive, Bashaw, Stettler, Caprona, Big Valley, Drumheller, and westward, near Olds, knowledge of the Devonian reef area has greatly increased. Numerous other wells have recorded the presence of the Devonian reefs in this general area and, while they were not productive, the general indication is that Devonian reefs are present over a much larger area than was previously thought, and certainly more Devonian fields will be found in central Alberta.

The light crude production obtained at Virden, Manitoba, set off a highly competitive 'lease play' in southwestern Manitoba and southern Saskatchewan, which was further stimulated by crude discoveries in North Dakota. Most significant were the greatly increased prices being paid for land in this area. Late in 1951, another discovery was reported twenty miles south of Virden, but no evaluation of this can be made at this time. These Manitoba discoveries have been in the Mississippian and could have considerable geological significance.

Gas discoveries, both in new areas and in extensions of known fields, have been very numerous and scattered, with the largest concentrations in the Edmonton area, at Bremner, Legal, and Sturgeon; in the northern Alberta areas, at Whitelaw, Tangent, Little Smoky, and Valley View; and in east-central Alberta, at Cessford, Sweet Grass, Maygill, and Many Islands. The Peace River discoveries were highlighted by the uncontrolled 'blow' at Hudson's Bay Union Tangent well, which was reported to be flowing at 50 to 80 million cubic feet per day.

Most of the Peace River discoveries have been in the Triassic and Permo-Pennsylvanian zones. The light crude at Fort St. John was also from the Permo-Pennsylvanian and geologists are now studying the overall Peace River area in the light of these recent finds.

Several large gas discoveries were reported in Saskatchewan, which is particularly important as they are the first reported in that Province.

Small gas finds were reported in the Fort St. John area of British Columbia and in the Northwest Territories near Fort Providence.

The general picture of discoveries in Canada can best be illustrated by reviewing events of the first three weeks in December. 1951. Discoveries of oil and gas were made at the rate of more than one every two days, a record rate for Canada's petroleum industry. A dozen of these strikes were made in Alberta and one in Manitoba. Six of the Alberta strikes found oil (Drumheller, Skaro, East Edmonton, Coutts, East Campbell, and South Calmar), one a distillate bearing natural gas (Bonnie Glen), and five gas (Chancellor, Hairy Hill, Sullivan Creek, Wavy Lake, and Hanna). The Manitoba strike reported oil.

In reviewing exploration and development activities in 1951, and their results, one is, perhaps, most impressed by the rapidity with which the petroleum industry in Canada is coming of age. When we speak of coming of age of the industry, we naturally turn for comparison, to the highly developed petroleum industry of the United States.

Mention has been made of the recent rash of oil dissoveries in central Alberta and it has been noted that, while one or two of these may prove to be major fields, the rest appear at this stage to be relatively small. Being smaller fields, however, does not in any way detract from their importance: it is the smaller fields that form the backbone of the United States' tremendous oil production. In other words, individually the smaller fields may not contribute a great deal to the oil stream—together, they make a very substantial contribution.

Mineral right holdings in the areas of recent discoveries give even stronger evidence of this coming of age. In earlier days of Western Canada oil exploration, the few exploration companies at that time, with competition for leases not too strong, could secure leases over relatively large areas. Today, perticularly in the central Alberta region, this competition for leases is fierce and results of this competition are evident in some of the newly discovered fields, where no one company holds more than 20 per cent of the leases, thus paralleling the situation in the producing areas of the United States.

In the pioneer days, exploration drilling was spaced far apart to prove or disprove large areas. Today, in central Alberta, we find companies working, re-working, and re-studying data already secured, with resulting closer density of exploratory drilling. An example of this is the Wizard Lake discovery which, in 1951, indicated a major field less than 1½ miles from a dry hole drilled four and a half years ago. This type of exploration has evolved gradually but concrete results were starting to show only in 1951.

In the earlier days of oil exploration and development, most companies carried on their activities with their own men and equipment. By 1951, practically all drilling was being done with rigs owned and operated by drilling companies. There are now, in Canada, a number of service and supply companies prepared and qualified to do, on a contract basis, any of the multitude of jobs required in the oil fields. For example, one oil company has been carrying on an active exploration and development programme requiring up to eight rigs at one time and has been able to operate with not more than eight to ten people on its payroll, all the drilling, geological, engineering, and production work being done by outside firms. More and more of the specialized oilfield equipment required is being manufactured in Canada.

Production

As is shown in Figure 6, Western Canada's oil production in 1951 was approximately 65 per cent higher than in 1950, being estimated at just over 48 million barrels — an average of 132,000 barrels per day and a peak of 180,400 barrels per day. In Eastern Canada, production was quite uniform and averaged 600 barrels per day. Figure 7 shows the increase in production and development that has taken place over the past four years. The marked in-

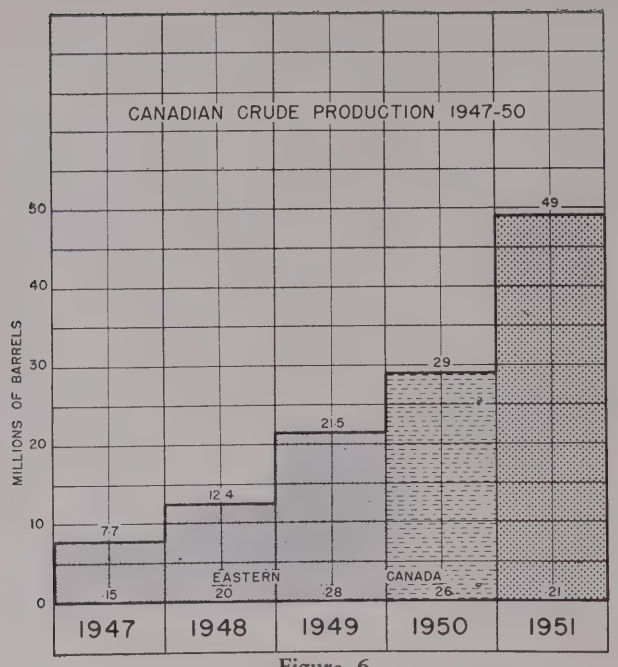

Figure 6.

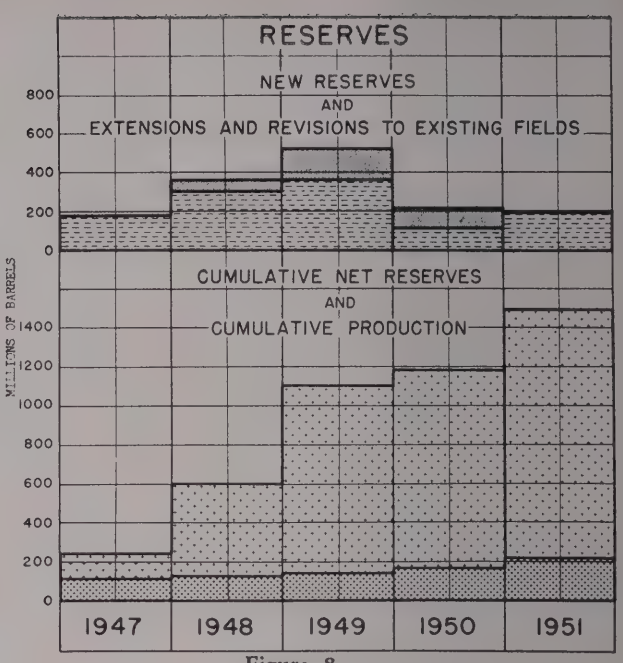

Figure 8.

crease during 1951 is mainly due to the large volume of Western crude now being transported by pipeline to Superior and thence by lake tanker to Sarnia, as well as to increased outlets to a small American refinery at Superior and to the widening of the area supplied by the new Winnipeg refinery.

Reserves

Estimates of the new reserves of crude that have been added during 1951 show a decided upswing from the 1950 figures. Some of this increase in reserves results from newly discovered fields and some from revisions to reserves of earlier known fields. As more data become available on producing fields, the reserve estimates are being constantly revised, and usually in an upward direction because the industry as a whole is conservative when estimating reserve possibilities of new fields.

On Figure 8, which is based on data compiled by the Western Canada Petroleum Association Reserves Committee, an endeavour has been made to show the estimated net remaining oil reserves at the end of each of the years 1947 to 1951, and to show what part of the increase has been due to new discoveries and what part to revisions. It is apparent that the Canadian oil industry is staying well ahead of requirements.

Reserve estimates on gas vary considerably, as witness the diversity of estimates of Alberta's reserves presented during the hearings on gas reserves. These estimates ranged all the way from 4½ to 13½ trillion cubic feet. All that can be said is that reserves are undoubtedly up considerably from the 1950 figures presented at a similar hearing, which ranged from 4½ to 7 trillion cubic feet.

	1948		1949		1950		1951	
Field	No. of Wells Dec.31	Ave.Daily Dec. Prod.	No. of Wells Dec.31	Ave.Daily Dec. Prod.	No. of Wells Dec.31	Ave.Daily Dec. Prod.	No. of Wells Dec.31	Ave.Daily Dec. Prod.
Redwater	2	550	296	12,390	753	35,820	898	49,470
Excelsior	-	-	1	50	22	1,040	32	1,690
Leduc-Woodbend	172	21,330	355	25,450	526	25,310	815	39,690
Acheson	-	-	-	-	6	410	55	3,570
Golden Spike	-	-	2	320	6	580	9	1,190
Joseph Lake	-	-	4	380	31	1,020	70	2,930
Stettler	-	-	3	290	29	1,190	46	2,700
Big Valley	-	-	-	-	4	290	22	1,210
Duhamel	-	-	-	-	1	-	12	650
Wizard Lake	-	-	-	-	-	-	8	1,600
New Norway	-	-	-	-	-	-	6	240
Caprona	-	-	-	-	-	-	2	120
Armena	-	-	-	-	-	-	15	360
T.Valley	293	11,300	296	10,100	298	9,350	298	7,800
Lloydminster Coleville, etc.	209	2,650	231	3,820	269	5,660	353	5,040
Others W.C.	131	1,790	146	1,980	203	1,150	310	3,620
Total W.C.	807	37,620	1,334	54,780	2,148	81,820	2,951	121,880
Eastern C.	-	540	-	770	-	730	-	580
Total	-	38,160	-	55,550	-	82,550	-	122,460

Figure 7.—Growth and development of oil production in Canada, 1948-1951.

Outlets

The outlets for Canadian crude underwent a great change during 1951 and announced programmes will bring still greater changes. Comparison of the Prairie crude production and the Prairie product demand for the years 1950 and 1951 will illustrate one phase of the change in outlets:

	Prairie Crude Production	Prairie Products Demand
	(barrels)	(barrels)
1950	28,800,000	28,200,000
1951	48,400,000	32,000,000
Increase	19,600,000	3,800,000

From the above it is evident that the prairie petroleum picture has altered substantially, with net exports reaching 16,400,000 barrels. These changes reflect the delivery of Western Canada crude to the central Ontario market *via* pipeline to Superior and tanker to Ontario refining centres, the completion of additional and enlargement of existing prairie refineries, and the deliveries of small amounts of crude to the American markets at Superior.

New refineries were completed at Edmonton (two) and Winnipeg during 1951 to increase prairie refining capacity by some 21,000 barrels per day. Additional pump stations were installed on the Interprovincial line, and increased storage was provided to allow for additional throughput. Two of the world's largest freshwater tankers, each designed for 115,000 barrels capacity, were placed in service in 1951 and transported some 5,000,000 bbl. each. A small refinery in Montana was dismantled, moved, and re-erected at Superior to provide an American outlet for some 4,000 barrels per day of Western crude. Late in 1951, small tanker shipments were made to a refinery in the Muskegon area.

As seen from Figure 8, there was a 13 per cent increase in the demand for products and this is in line with the similar overall increase in Canadian demand. One of the important items making up the 13 per cent increase in prairie products was the greatly augmented market for black oil, or heavy crude, stocks. Conversion of several hundred locomotives in Western Canada to oil burners, coupled with the 5-year contract for 2,000,000 barrels per year placed by the Canadian National Railways, stimulated production of heavy products. Three new building-products plants (roofing and building papers) recently constructed at Winnipeg, Lloydminster, and Edmonton, together with increased Western highway surfacing programmes, have created additional demands for heavy crudes.

The general crude outlet will be greatly increased in 1952 and following years by projects commenced or announced during 1951. The most important of these was the granting of a permit for the construction of 693 miles of oil pipeline from Edmonton to Vancouver. The contracts for this line have already been awarded and construction is expected to be completed by the fall of 1953. The line as presently designed, with pump stations at Edmonton and Kamloops, will handle 75,000 barrels per day and initial deliveries are expected to be 35,000 barrels per day.

During 1951, construction was commenced on one new refinery at Froomfield, Ontario, designed to process 20,000 barrels per day. New refineries have also been announced for Winnipeg and Vancouver. Extensions have been commenced or proposed to existing refineries at Sarnia (increase 13,000 barrels per day), Moose Jaw, Edmonton, and Vancouver. The new refinery and expansion proposed for the Vancouver area are expected to increase the overall capacity there by 60 per cent, to 50,000 barrels per day.

For transport of additional western crude from Superior to the Ontario market, construction of two new tankers of 115,000 barerls capacity was commenced in 1951. One will go into service in the spring of 1952, the other later in the year. At the same time, increased tankage is being provided both at the eastern end of the Edmonton-Superior line and at Sarnia, and the throughput of the line itself will be stepped up.

A pipeline to move refined products from Sarnia to London, Hamilton and Toronto was commenced during the year. Completion of this line will extend eastward the territory which can be supplied by Western Canadian crude. Plans were announced also for construction of a products line from Montreal to Toronto, to bring products from Montreal to the Toronto area. This line could have a major effect on the transportation of refined products in the eastern Ontario region.

Outlets for gas have not materially changed. A permit to allow the export of limited amounts of gas from southeastern Alberta to Great Falls, Montana, as a defense measure, was issued by the Alberta Government. This permit provides for a maximum delivery of 7½ million cubic feet per day from the Pakowki Lake field, and a 75-mile pipeline has been constructed to bring this gas to lines that were already installed in Montana. Other outlets for Alberta gas, to the West Coast, to central and northwestern Washington, and to Saskatchewan, Manitoba, and Ontario, are all dependent on permission to export gas from Alberta. A hearing on gas reserves was held in the fall of 1951 but no decision as to whether the Alberta Government considers them sufficient to allow export is expected until after February, 1952.

The completion of a gasoline plant at Jumping Pound and the construction of a line from this field to Calgary and also to the cement plant at Exshaw and to the town of Banff, allowed additional gas reserves to be utilized.

In Eastern Canada, the demand for gas continues to greatly exceed the supply.

Petrochemical Industries

Probably the most significant development in the Canadian petroleum industry in 1951 was in the field of petrochemical by-products.

Plants for the manufacture of such products had already been established during the late war years and in the immediately following years, as for example the synthetic rubber and plastic plants in Sarnia and the synthetic ammonia plant in Calgary. In 1951, announcement was made of plans for, or actual commencement of, numerous plants in both Eastern and Western Canada.

In the Montreal area, three plants are proposed for manufacture of acetone, phenol, isopropanol, and other products from cracked refinery gases.

In Western Canada, two plants for the production of sulphur from natural gas are under construction in Jumping Pound and Turner Valley, with a total capacity of 50 to 70 tons per day.

A 17-million-dollar plant is proposed for the Edmonton area for the production of nickel, the synthetic ammonia required in the process being produced from natural gas.

A large plant in Edmonton, to cost between 40 and 50 million dollars, for production of celanese and a large number of chemical by-products, is under construction and expected to be in operation in 1952. As raw material, this plant will use butane and propane obtained from local refineries and oilfields, as well as large quantities of natural gas. Another Edmonton plant, for the manufacture of polythene, has been announced. It will cost 12 to 15 million dollars and will use as its raw material the ethane extracted from gas produced in the Leduc field.

The importance of the petro-

chemical industries as a stabilizing influence on the market for petroleum and related chemicals has been well demonstrated in Texas and other areas. It is anticipated that they will be equally important to the petroleum industry in Canada.

These new petroleum outlets, as well as the new pipelines, tankers, and refineries, all planned or commenced during 1951, will naturally give a great impetus to exploration and development activities. While the exploration and development programmes of last year were the most extensive in the history of Canada's petroleum industry, new records will have to be achieved in practically all phases of the industry in 1952 and future years, if the necessary gas and oil is to be made available for the widening markets that may be confidently anticipated.

Progress in Coal Technology: Coal Preparation*

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INTRODUCTION

GENERALLY, the necessity for mining coals of higher impurity content, the drive for lower costs by intensive mine mechanization, changing market requirements, and the increasing competition of oil and natural gas, have continued to result in an upward rise in coal preparation, and have necessitated a more intensive study of coal preparation problems.

Increasing adoption of continuous mining equipment, for example, is introducing new problems in coal preparation. Limited experimental work in the United States (1) shows that the effect of the continuous miner on size distribution varies with the characteristics of the seam. In some seams, the fines problem is lessened, in others it is greatly aggravated. Generally there is a decrease in the relative amount of the top sizes (plus 2 in.) and a substantial increase in the intermediate sizes (down to about 1/8 in.). Contrary to expected results, where continuous miners with vertical-moving cutting heads are used, an increase in impurities in the slack sizes does not necessarily occur where draw-slate is taken. This is because the draw-slate tends to be kicked out in lump form. However, if a seam contains bands and lenses of impurities they are bound to be in the mined product. In addition, because of the use of from 3 to 12 gallons of water to allay dust in continuous mining, the slack sizes are wet, sometimes containing 12 to 15 per cent water, a factor which enhances problems of screening and drying to meet market requirements.

These changes in the coal as mined will, it is anticipated, require certain plant changes. Coarse-coal picking operations may be eliminated or reduced. Screening of the fines may have to be done wet and, for those mines where the fines are greatly increased, additional facilities will be needed for water clarification, where wet cleaning methods are employed. In addition, the capacity requirements of certain equipment will be greatly different for continuous-mined coal as compared to conventionally-mined coal.

GENERAL COAL PREPARATION

Heavy Media Coal Washing

Although sales records in the United States (2) show that the Baum jig still appears to be the favourite, with approximately 50 per cent of new installed capacity, heavy-media plants are increasing in popularity, accounting for about 27 per cent of new installed capacity. Whereas, in 1948, there was only one heavy-media plant in operation in the United States for coal preparation, with a feed rate of 250 t.p.h., today there are 45 plants in operation or under construction with a combined feed rate of 8,000 t.p.h., and in Western Canada there are three plants.

The situation in Great Britain is very much the same. Whereas about twenty years ago there was only one heavy-media plant in commercial operation, the Chance process, today there are about 70 plants in operation using suspensions of sand, magnetite or mill scale, barytes, froth-flotation tailings, and crushed shale.

As indicating the possibilities due to the development of improved cleaners, it is of interest to note that recently, in North America, several heavy-media plants have been established mainly for reclaiming coal from waste banks. In the Pennsylvania Anthracite field (3), one company, using a Wemco Mobil-Mill drum-type heavy-media separator, is reclaiming about 12 per cent of a refuse dump as clean coal floating at 1.70 specific gravity. In this plant, operating at 150 to 175 t.p.h., 2¼ in. by 3/16 in. material is cleaned in the heavy-media unit, whereas the 3/32 in. by 3/16 in. fines are cleaned on two Deister tables.

New developments in the field of heavy-media separators are stressing reduction in capital cost as well as reduction in maintenance, with special reference to the increasing demand for smaller units.

In the Vögel process, a full-scale plant for which was constructed recently in Germany, the coal and magnetite medium travel together in a troughed belt. The floating material (clean coal) is removed at the discharge end by a knife and the sinks are delivered from the end of the belt. The process, as presently in operation, is limited because it has been designed for sized coal, but it is claimed that coal down to 1 mm. size can be satisfactorily washed. The size of the belt varies with the size of the coal. It must be 2½ to 4 times as deep as the largest particle, with, for example, a length of 20 feet for cobble size and 60 feet for coal below ½-in.

More recently, a similar belt heavy-medium separator has been described by Schönmüller (4) which, it is claimed, is capable of washing sizes from about 3.2 inches to 28 mesh in a single separation process, the coal being first de-sludged. A reduction of about 20 per cent in capital cost is claimed.

In the United States, both the Akins and Hardinge type classifiers have been adapted for heavy-medium separation, using a suspension of magnetite in water as the heavy medium.

The Hardinge counter-current (5) two-product unit consists of a gently sloping, rotating drum which is twice as long as its diameter. The inner surface is equipped with spiral flights that increase in height and decrease in pitch from its feed end to the refuse discharge end, where the refuse is collected in a narrow cylinder and elevated by lifter flights to a refuse discharge chute. Raw feed and media are introduced at the low end of the drum. Settled

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(1) For references, see end of paper.

refuse is carried toward the upper end by the spiral flights, while floating clean coal is carried by the medium to the clean-coal discharge at the low or pool end of the unit. Coal separators of this type are available in seven sizes, from an 18-in.-diameter laboratory unit to a 10-ft.diameter machine, the latter with a capacity of 500 t.p.h. of feed. Mechanical advantages are said to be low horsepower requirements, low medium requirement, low medium-to-coal ratio, and low maintenance cost. An additional advantage is that the separator can be started under full load.

The Akins *HMS* separator, which originally was employed as a two-product unit, has now also been commercially developed to a three-product separator (6), cleaning about 200 t.p.h. of 7 in. by ¼ in. West Virginia bituminous coal. The three-product vessel is divided into two parts, a horizontal or pool section and an inclined spiral. The sink-float separation takes place in the pool section and the sink is removed by the spiral conveyor. The vessel is equipped with a side feed and side discharge for the float coal. The float material flows directly across the width of the vessel near the end of the pool, whereas the midlings are carried by the submerged screw conveyor to the front part of the pool and removed by a cross-flow of the media from the overflow of a media head tank. As in the case of the Hardinge machine, it is not necessary to empty the vessel during periods of shut-down as the turbulence caused by the screw conveyor and by media entering the pool puts the magnetite back into suspension.

In the field of drum and wheel-type heavy medium separators, Simon-Carves, Limited, have developed a rather novel vessel, the *Simcar* washer (7). The separator has a bath which is cruciform in plan and contains the lower arcs of two wheels interlaced at right angles, one for coal and the other for refuse. Each compartment of a star feeder delivers a controlled quantity of coal to one compartment of the coal wheel. Medium is introduced to the bath immediately below the feeder and, as the raw coal is carried down below the surface of the medium, it begins to separate. Further travel of the wheel brings the compartment over the refuse wheel, into which the separated refuse falls, while the floating coal is carried forward by the coal wheel to the clean-coal discharge.

Simon-Carves also claim to have developed a more efficient method of reclaiming the magnetite from the dilute medium. In order to obtain higher settling rates and use smaller apparatus, the magnetite is magnetized while the liquid is in a non-turbulent condition. The medium-separator is in the form of a small settling cone. The dilute medium is introduced through a pipe discharging against the underside of a revolving non-magnetic dic and a specially designed electromagnet is located over the disc immediately above the flared mouth of the pipe. Particles of the magnetic medium are held by the magnet to the underside of the revolving disc until they are carried out of the magnetic field, when they rapidly sink to the bottom of the cone. Slimes and excess water are carried over the launder of the cone, while the magnetite is densified, making it possible to draw medium from the base at a density greater than that employed in the washing bath.

Although, in the past, all the heavy-media separators used in the United States were designed in that country, recently the Dutch Tromp (8) process, using a magnetite medium, has been introduced. The first plant of this type to be installed in the United States was at the Old Ben Coal Corporation's Mine No. 32, in southern Illinois. The rated capacity of the plant on a 2-shift basis is 9,200 tons. The 6½ in. by 1½ in. size is processed at a specific gravity of 1.36 in two two-product vessels (300 t.p.h.), and the 1½ in. by 5/16 in. (285 t.p.h.) in two two-product vessels at 1.43 gravity. The plus 1 in. rejects from these are processed in a fifth two-product vessel at a gravity of 1.60. The floats from this latter are crushed to free coal from laminated bone and are then re-circulated to the raw-coal screens for re-treatment. A three-product vessel is also available.

In so far as the type of medium is concerned, the present tendency appears to be toward the use of magnetite, according to Chapman (9). The Dutch State Mines processes using froth-flotation tailing or crushed shale, media admittedly cheaper than barytes or magnetite, cannot, however, provide very high effective densities in the medium without it becoming too viscous. When the medium becomes too viscous, the free movement of particles below 1 in., and of particles with specific gravity close to that of the medium, is seriously impeded, with a reduction in capacity and efficiency. The highest density obtainable for effective wahing in a static bath with such media is claimed to be about 1.55, and with upward currents, about 1.65.

In using magnetite with a density of 4.8 to 5.0, media with densities from about 1.35 to 1.75, with relatively low viscosity, can be produced because of a lower concentration of suspended solids. The great advantage of magnetite, of course, is that it can be recovered and concentrated by magnetic means.

In connection with magnetite as a medium it is of interest to note the magnetic process developed by Simon-Carves, Limited (7), for obtaining low-cost magnetite from blast furnace flue dust, which contains from 20 to 60 per cent of magnetic material. For a plant treating 250 tons per week of flue dust containing 50 per cent magnetic material, the total preparation cost, exclusive of raw material charges, is estimated at about $3.63 per ton, rising to about $5.10 per ton if the magnetic yield is only 30 per cent. This compares with a cost in Great Britain of imported natural magnetite of $20.00 to $24.00 per ton. The medium separated from blast-furnace dust may be too coarse and is milled to 95 to 98 per cent below 200 mesh (BSS), using a rubber-lined ball-mill.

Baum-Type Jigs

As stated before, the Baum jig still appears to be a favourite wet washer when the cleaning job is not too difficult and when separations below 1.60, or possibly 1.50, are not required. The Baum jig has an advantage over heavy-media plants in that capital cost is lower, and equipment is available for a very large range in outputs from the smallest to the largest mines.

The system of 'back-stroke jigging' introduced by B. H. Bird, of the Jeffrey Manufacturingg Company, has apparently revitalized the interest in jig washing, especially as regards small coal. Instead of the usual air-pressure of 2½ lb. per sq. in. applied to the jig at a rate of 45 pulsations per minute, Bird has used pressures of 3½ to 4 lb. per sq. in. applied at as low a rate as 22 pulations per minute. By using a special type of valve which admits air rapidly to the wash-box, the bed is lifted bodily from the bed-plate. This is followed by an expansion period that enables the bed to open out. The exhaust stroke

is then rapidly applied, giving a strong suction, which causes the heavy particles to fall rapidly. This is a complete reversal of previous jigging practice, which was based on the assumption that stratification occurs during the upward phase of the water currents. Improvement in capacities and efficiencies are claimed and, because less circulation water is used, there may be a reduction in slurry. Recent experimental application of the Bird cycle in Great Britain (9) appears to confirm the improved results obtained in the United States. In one plant, introduction of this principle will enable the colliery to feed an additional 50 t.p.h. to their jig, increasing the capacity to 200 t.p.h.

A new type of Baum jig (10) designed by the Haworth Engineering and Manufacturing Company, Birmingham, Ala., and recently installed in a cleaning plant in Tennessee, contains some rather novel features. In the small 2-compartment, 10-cell, air-operated jig, the screen plates are independently adjustable over each two cells, which permits the plates to be pitched rather steeply at the entrance to the refuse draws, and flattened toward the centre of the jig. It is claimed this allows for more accurate control of the refuse bed. The refuse draws are reciprocating plate feeders operating at slow speeds and short strokes. Both the refuse draws and the elevators are of the variable-speed type, being operated by d.c. motors through selenium rectifiers and automatic timing control relays. The refuse elevators speed-up and slow-down automatically with the refuse draws, in accordance with a float control on the refuse bed, a 10 to 1 speed variation being accomplished in a fraction of a second, thus holding the depth of the refuse bed constant.

Standardized Combination in Preparation Plants

Operation of the coal mines in Great Britain under the National Coal Board has enabled the development of a standard design of preparation plant (11). By this means it has been possible to reduce the work required in contractor's drawing offices, lower the costs of manufacture, and reduce by almost one-half the time taken for the erection of a cleaning plant, and it enables multiple units of wash boxes, screens, etc., to be held in stock. A standard Baum plant was devised with capacities of 50, 100, 150, and 200 t.p.h. and incorporated in a standard building. A froth flotation plant was also standardized so that it could be incorporated in the standard building.

A further step in standardization was the development of a simple Baum washing plant suitable for small mines (50 t.p.h.) with coal not exceeding 4 in. This plant dispenses with a settling cone, using a dredge tank instead. Two designs are available, one for two-products and the other in which middlings are crushed and re-circulated.

In addition, there was developed a combination heavy-medium-Baum plant with a capacity of 220 t.p.h. of 8-10 in. coal, using the Barvoys heavy-medium process.

Mechanical Cleaning Versus Hand Cleaning

Although mechanical cleaning of large coal is no doubt superior to hand-cleaning, there is still much discussion concerning the relative benefits and merits of each. Generally, the opinion is that hand-picking is unsatisfactory, and the trend is toward mechanical treatment. However, under certain conditions, hand-picking intergrown pieces may be economical in reclaiming near gravity particles for crushing and washing (13).

Chapman (14), in discussing some experiments recently conducted on a relatively large scale in Great Britain, draws the following conclusions: (*a*) in so far as operating cost is concerned, there are no general grounds for cleaning coal below 3 in. by hand; (*b*) the cost of cleaning by hand rises very sharply with increasing quantities of reject, including inferior coal; hand-cleaning of cobble size (*e.g.*, 6-3 in.) has some attraction if the amount of reject is only about 10 per cent; (*c*) mechanically cleaned coal is of more uniform quality than hand-cleaned coal, and for the same recovered tonnage it will have a lower ash content.

In connection with hand-cleaning, the circular picking-table (14) developed at Gedling Colliery in England has certain advantages over the usual type of picking belt. With this picking table, no lifting is required, the pieces for removal from the surface being slid over a smooth surface to a position where they can be scraped away by a stationary plough. By reducing fatigue and eliminating the time occupied in lifting, tests at Gedley mine on coal over 5 in. have shown that a weight of 25 tons/man-shift can be sorted on the table into different products, which is double the amount for the usual type of picking belt.

Preparation of Fines

During 1951, wet tables, air cleaners, and flotation units made additional gains in fine-coal cleaning, and impetus was given to predrying of coal prior to air-tabling, because of the improvement in cleaning. In North America, especially when the coal is difficult to clean, wet tables were almost exclusively installed for fines passing the ½-in. screen. In several of the states of the United States, under the pressure of the programme to prevent stream pollution, froth flotation has been much discussed and some new plants were installed. In view of the very high cost of coals and the increasing ash content of many slurries in Europe, increasing attention is being paid to flotation methods, as well as to flocculation.

Heavy-Media Separation

Notable among the advances in cleaning fine coal is the adaption in Belgium by Charbonnages d'Hensies Pommeroeul (15) of a Wemco Mobil Mill heavy-medium separator to washing coal down to 1 mm. This experimental work has been fully presented by J. Baudry (15), and laboratory work has apparently shown that difficulties due to viscosity do not appear in the treatment of sizes 1 to 10 mm. if a suspension of magnetite in water is used for densities between 1.4 and 1.7, Stokes' law applying only to particles smaller than ½ mm. The extensive experiments indicated that: (1) for the same washing equipment, a reduction in size reduces the hourly capacity of the plant and increases the magnetite consumption; (2) removal of minus 1 mm. slimes and wetting the coal before introduction to the separator avoids surface-tension problems and improves the ultimate recovery of magnetite; (3) to reduce magnetite loss, the size of the drainage and rinsing screens should be adjusted so that the thickness of material bed does not exceed 1.2 to 1.6 in., and rinsing should be carried out with large quantities of high-pressure water. Furthermore, a thickener should be provided to receive the rinse water, thus permitting thorough rinsing without increasing the number of magnetic separators required, these latter (usually two) being operated in series. Experimental runs over a period of six months cleaning certain 30.8 per cent ash fines yielded clean coal varying between 5.8 and

6.9 per cent in ash and refuse with between 72 and 74 per cent ash.

Air-Separation

Under certain conditions, high-ash fines may be separated from slack by means of an air-separator (16). At the Brilliant mine of the Kemerer Coal Company, Frontier, Wyoming, where the 0-28 mesh fines are relatively high in ash, a 14-ft. Sturtevant air-separator with a rated capacity of 40 t.p.h. feed is used for separating the minus 28 mesh dust from the 3/16-0 in. slack; the 28 mesh-3/16-in. size averaging about 8 per cent in ash, and the 0-28 mesh rejects containing up to 16 per cent ash. Although small variations in size-consist of the feed do not limit operations, surface moisture has a deleterious effect and provision was made for adding warm make-up air. Surface moisture, also, increases the danger of freezing in the separator and to offset this, electric vibrators were installed in the outer cone of the separator to break up the agglomerations of frozen fines.

Froth Flotation

The Automatic Coal Cleaning Company, Limited, Carlisle, England, has developed a new type of froth-flotation cell (17). Instead of the vertical shaft usually employed, two opposed propellers on a horizontal shaft near the bottom of the cell project into the cell two opposing streams of mingled air and pulp, resulting, it is claimed, in a degree of mixing and aeration previously unapproached without the use of pumps and blowers. The tiny bubbles rise through a quiescent zone to form a dense mat over the entire surface, which is discharged from one side of the cell by continuously moving blades which traverse the entire surface of the cell.

As mechanization and dust suppression methods underground are extended, so the treatment of fines and slurries presents increasing problems. Coal preparation, where wet fines have become a serious problem, has been materially aided by the fact that recent experiments have shown that it is possible to wet-screen coals, on slightly modified vibration screens, at 1/16 in., as efficiently as screening fines dry at the same mesh size (18). These results, together with the fact that coals of size 1/16-0 in. can be treated by froth-flotation, have encouraged in England the cleaning of run-of-mine coal in two stages, by a heavy-medium washing plant for the plus 1/16 in. coal and flotation for the minus 1/16 in. A large plant, using such a two-stage process, with a capacity of 450 t.p.h., has been constructed at Mansfield Colliery, East Midlands division. In this manner, it is considered that large economies in capital expenditure and in running cost will be effected.

Slurry Handling

Slurry, defined as that material, minus ¼ mm. in size, which contaminates the circulating water in any wet-washing system, cannot be recovered by normal screening methods, but is now being successfully recovered by flocculation (18). This has not only aided in clarifying washery effluents but has facilitated the introduction of closed circuits in the washery system. The method is also applicable to the recovery of fines with moderate ash content. There are apparently no coal washing effluents which cannot be flocculated. Generally speaking, after dosing the circulating water carrying the suspended solids with lime to adjust the *pH* value to about 9, a suitable starch reagent, such as Dryfloc, Flocgel, Tragacene or Unifloc, is added to the effluent at about 0.5 to 3.0 pounds of reagent per thousand gallons of effluent treated. The flocculated solids, after being settled out and thickened in a circular tank with launder, are usually dewatered in a vacuum filter. Many benefits accrue from operating a cleaning plant with fairly clean circulating water. The ash content of the smaller sizes down to ¼ mm. is improved, the percentage discard of coal in the refuse is materially reduced where automatic controls are in operation, abrasion on pumps, valves, pipe lines and plant in general is considerably reduced, and the moisture content of the washed products, especially the fines, is materially decreased.

Recently, increased attention has been devoted to closing preparation circuits with vortex cyclone thickeners alone or by cyclones followed by filters (19). The usual Dutch State Mines cyclone has a diameter of about 15 in., an inlet opening of about 2 in. diameter, and an outlet at the apex of 1 in. diameter. The washery water is admitted tangentially at a speed of about 20 ft. per second and at a pressure of about 30 lb./sq. in. It is probable that three or four such cyclones would be as effective as a 30-ft. thickener, without the added expense of flocculating agents (9), for treating about 20,000 gal. of washery water per hour. The cyclone has the disadvantage that it would not remove the ultra-fines from the circuit.

Dust Reduction

A step forward in reducing dust underground due to drilling, without the necessity of using water, which results in troublesome wet fines, has been made by the introduction of special drills associated with dust collectors. In the *Colliery Guardian* of Sept., 1951, two kinds of British drills of this type are described. The Hugh Wood & Company, Limited, Gateshead-on-Tyne, dustless drill is devised as follows: A metal sleeve, through which the drill rod operates, is fitted to the drilling machine and is connected by hose to a suction tank. In this dust receiver compressed air is admitted from the main and discharged through an ejector, thus creating a drop in pressure in the reservoir. Holman Bros., Limited, have also produced what is claimed to be a completely dustless rock drill, the *'Dryductor'*, which extracts the dust as it is formed in the hole and ejects it through a sleeve into a cyclone separator, bag filter, or combination, suction being created by a fan. By such means it is possible not only to reduce coal dust hazards in mining but to reduce also coal preparation problems resulting from wet fines and slimes. In the field of dust collection, it should be noted that there has been an increased use of wet-type collectors. One such type is the 'Air-Tumbler', manufactured by the Dust Suppression Engineering Company in the United States (20). In this machine, dust-laden gases are forced through a helical path of horizontal axis and encounter the water surface at the bottom of each turn. Centrifugal force throws the particles out in contact with the water or against the wet sides. Sludge from the tumbler is pumped to a sludge tank. At the Leatherwood mine, the overflow from the sludge tank is clarified by a 40 in. by 60 in. Bird polisher, but other equally suitable methods of clarification may be used, such as cyclone thickeners and vacuum filters.

Drying and Dewatering

As the trend toward the cleaning of fine coal by wet methods increases, and as the necessity for washery water clarification increases, the thermal drying of fine coal, preceded by mechanical dewatering, is becoming very important in coal preparation.

As a result of a survey, conducted under the sponsorship of Bituminous Coal Research, Inc., concerning the performance characteristics of various thermal dryers manufactured in the United States (21), the flash dryer appeared best adapted to the drying of fine coal. However, because of the large dust collecting systems required to reclaim the fine coal, and the relatively poor thermal efficiencies caused by the large amounts of sensible heat vented with the exhaust gases, the Battelle Memorial Institute inaugurated an investigation to design a fluidized-bed dryer which would overcome the disadvantages of the flash-dryer (22). The experimental pilot-plant test work indicated that high thermal efficiencies require very high heat inputs, and it is believed that gas with less than 2 per cent oxygen and with inlet temperatures of about 2,000°F. are feasible and capable of giving a dryer capacity of about 1,700 pounds of dry coal per square foot per hour with a feed containing 25 per cent moisture. A commercial unit has not yet been established.

The reciprocating-screen type Vissac Updraft Pulso fines dryer is finding increasing favour in North America. One of the first installations, at a mine in the State of Washington (23), drys 0-¼ in. washed coal, preliminarily dewatered in a centrifuge to 8 to 10 per cent surface moisture, down to 1 to 2 per cent surface moisture. The 45 t.p.h. drier includes a single 6 ft. by 12 ft. deck fitted with 1½ to 2 mm. wedge-wire screen. The hot gases are blown in at 525°F. to 650°F., and the exhaust gases exit at 90°F. to 250°F., the fines in the exhaust gases being removed by two 7-ft. cyclones. This fines dryer (marketed in the United States by the McNally Pittsburg Mfg. Corp., and by G. A. Vissac, the designer, through the Riverside Iron Works, Ltd., in Canada) has recently been applied to the drying of slack coal, with varying moisture content, as it comes from a strip pit in the Minto field of New Brunswick.

Miscellaneous

Sizing of Commercial Grades of Coal

With a view to standardization in the size of graded products (12) which will be of benefit to the consumer, the manufacturer of screens, and the preparation engineer, the National Coal Board implemented the proposed system of size classification outlined in the *Report of the Coal Grading Committee of the British Colliery Owners' Research Association*, published in 1946. In addition to 'Large Coals' having no upper size limit, the report recommended seven main size groups with the following typical screen sizes (in. rd. hole):

(a)	Large Cobble	6-3 in.
(b)	Cobbles	4-2 in.
(c)	Trebles	3-2 in.
(d)	Doubles	2-1 in.
(e)	Singles	1-½ in.
(f)	Peas	½-¼ in.
(g)	Grains	¼-⅛ in.

Although no undersize tolerances were included, an upper and lower size limit for each grade was indicated. The material under 1/8 in., called 'smalls', has no lower size limit but may be divided into classes according to their fines content.

Generally speaking, it should be noted that, both in North America and Europe, the consumer trend is towards smaller sizes of coal. In view of the fact that introduction of mechanization in mining tends to increase the proportion of smaller sizes produced, the increased utilization of smaller sizes is a boon to the coal industry.

Carbonized Briquettes

The production of artificial smokeless fuels for the domestic market by the carbonization of briquetted raw coal is now well established in Great Britain as a result of the development of the Phurnacite process by Powell Duffryn, Ltd. (24, 25). Although only one plant is in existence, at Aberaman in Wales, the National Coal Board has recently extended it to 1,200 tons per day. The process consists of briquetting low-volatile, non or poorly caking coals with coal-tar pitch and subsequently carbonizing the product at low temperature. The carbonizing equipment, designed by the Disticoke Company, Limited, Paris, consists of two 40-oven batteries, the ovens being of the recuperative type, heated with re-cycled oven gas. The ovens have parallel walls, vertical flues, and inclined soles, have a capacity of just over two tons, and the carbonizing time is about four hours. Application of the process to other coals is anticipated. Recent experiments carried out on Scottish low-rank coals to produce large carbonized ovoids for cupola use in iron foundries have proved technically successful.

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Theory and Practice of Roasting Sulphide Concentrates*

With Special Reference to Canadian Gold Ores and Concentrates

By L. E. DJINGHEUZIAN†

(*Ottawa General Meeting, January,* 1952)

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Introduction

IN 1940, in a series of articles, M. W. von Bernewitz(1) reviewed in detail roasting of gold and silver sulphide ores and concentrates. Since that time some significant developments in roasting have taken place, and a number of papers have appeared in the Canadian technical literature, dealing severally with the roasting of a particular ore or concentrate.

The purpose of this paper is not only to review some of these Canadian papers and bring the information on Canadian roasting practices to date, but also to analyse the mechanism of roasting. Hence, it is proposed to start with a study of physico-chemical factors fundamental in roasting, the knowledge of which, by creating a sound basis for a proper roaster design, makes good roasting possible.

Physico-Chemical Factors

Roasting can be defined as heating sulphide ores in presence of air below fusion points of the constituent minerals, at such temperatures as to convert metallic sulphides to oxides. Hence, roasting is a chemical process.

The object of roasting is, by changing, chemically, mineral constituents of sulphide ores, to make the economic recovery of the desired constituent metal by a subsequent standard treatment possible. For instance, pyritic ores, containing gold in sub-microscopic form which no amount of grinding would release, are roasted in order, by oxidizing the sulphides, to open up their structures to such an extent as to enable cyanide solutions to come into contact with the gold and dissolve it. The production of porous calcine is, therefore, the object of roasting of refractory auriferous sulphide ores or concentrates. Such a porous calcine may be defined as a desirable calcine, and the roasts which produce desirable calcines can in their turn be termed desirable roasts.

In the conversion of sulphides to oxides, several chemical reactions take place. New compounds, both solid and gaseous, are formed and this in addition, depending on thermodynamic conditions prevailing during the roast, results in some side reactions taking place. Some of these reactions are beneficial and the others deleterious to production of desirable calcines. Some of these reactions, depending on variations of temperature or partial pressure of gases surrounding ore charge, are reversed, and others proceed forward at an accelerated rate. The following typical equations illustrate the point:

$$4FeS_2+11O_2 \longrightarrow 2Fe_2O_3+8SO_2 \quad (1)$$

$$2SO_2+O_2 \rightleftarrows 2SO_3 \quad (2)$$

$$Fe_2O_3+3SO_3 \rightleftarrows Fe_2(SO_4)_3 \quad (3)$$

The partial pressure of any gas is proportional to its relative volume in the mixture of gases. Suppose the SO_3 content in a mixture of nitrogen, oxygen, and sulphurous oxide is 3 per cent by volume. Then, at normal atmospheric pressure, the partial pressure of SO_3 will be equal to 23 mm. ($= 0.03 \times 760$) of mercury. At this partial pressure of SO_3, the critical temperature at which equilibrium will obtain in equation (3) is 553°C. (1,027°F.)(2). Now, if the partial pressure of SO_3 drops or if the temperature is raised above 1,027°F. the reaction will go to the left and more Fe_2O_3 will be formed. Thus, the control of the reaction requires the control of both variables.

The above example shows that in obtaining desirable roasts, the fundamental requirement is the control of chemical reactions, and from the above discussion it is obvious that the control of chemical reactions in roasting is effected through the control of temperatures and partial pressures of gases formed during roasting. The control of partial pressures of gases in the direction of removal can be effected by scrubbing SO_2, As_2O_3, etc., gases at suitable points in roasters.

Research done by Australian investigators(3) on roasting of Kalgoorlie auriferous pyrite concentrates, and particularly that by A. F. B. Norwood(4), is outstanding. They established that, to obtain calcines with maximum porosity, a low roasting temperature is essential during the elimination of the first atom of sulphur. Norwood states further that "a high temperature at any stage before the elimination of practically the whole of sulphide sulphur was harmful. This effect decreased as the sulphur content of the roast was reduced. The effect, however, was far less noticeable after the first atom of sulphur had been expelled from the pyrites than before this point".

Having arrived at a conclusion that in obtaining desirable calcines the roast was the controlling factor, Norwood proceeded to determine the chemical reactions occurring in laboratory and plant roasts at different temperatures.

Samples were taken at various points along the hearth of the roaster (Edwards) and these, together with the products of laboratory roasts at various temperatures, were analysed and their compositions plotted in a series of curves.

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(1) For references, see end of paper.

The study of these curves showed that high initial temperatures of over 650°C. (1,200°F.) favoured the formation of pyrrhotite, whereas temperatures between 450° and 550°C. (840° and 1,020°F.) favoured direct roasting of pyrite to magnetite. The temperatures were measured in the roast with a bare wire thermocouple. It was shown further that with the initial roasting temperature sufficiently high, no oxidation of pyrite took place and pyrrhotite alone was formed, which subsequently oxidized. Another important reaction shown by the curves was that, when the roast was approaching completion, much (and in some cases all) of the hematite formed during roasting was reduced to magnetite, which was re-oxidized again as soon as sulphur elimination was complete. This change was also effected by temperatures. Thus, it is felt by the writer that Norwood's investigations definitely established the necessity of temperature control in roasting of Kalgoorlie pyrite concentrates.

The reactions deduced from these curves were checked by series of roasts both in sulphur dioxide atmosphere and air by Norwood, and were summarized as follows:

The initial reactions:

$$4FeS_2+11O_2 \longrightarrow 2Fe_2O_3+8SO_2 \quad (1)$$
$$nFeS_2 \longrightarrow Fe_nS_{n+1}+(n-1)S \quad (2)$$
$$FeS_2+3O_2 \longrightarrow FeSO_4+SO_2 \quad (3)$$

At a later stage:

$$3Fe_nS_{n+1}+2nSO_2 \longrightarrow nFe_3O_4+(5n+3)S \quad (4)$$
$$4Fe_nS_{n+1}+(7n+4)O_2 \longrightarrow 2nFe_2O_3+(4n+4)SO_2 \quad (5)$$

When most of the sulphides have been oxidized:

$$6Fe_2O_3+2SO_2 \longrightarrow 4Fe_3O_4+2SO_3 \quad (6)$$

Finally, when oxidation is complete:

$$4Fe_3O_4+O_2 \longrightarrow 6Fe_2O_3 \quad (7)$$

Some ferric sulphate is also formed, at the same stage as reaction (6), especially if the finishing temperature of the roast is low:

$$Fe_2O_3+3SO_3 \longrightarrow Fe_2(SO_4)_3 \quad (8)$$

The last reaction indicates that the roast should be finished at a higher temperature in case it is desirable to convert any possible ferric sulphate present to ferric oxide.

When the above chemical reactions take place, gases are involved and these gases cause changes in partial pressures and volumes. Evidently, if the pressure of gas or gases forming equals or exceeds the opposing partial pressure of the same gas or gases already formed, the reaction goes to the right. Hence, at each location in the roaster, each chemical reaction must be favoured by a set of conditions of temperature and partial pressures which must be kept constant at that particular location. Any change in one of these conditions might either reverse the chemical reaction or cause another reaction to start taking place. Hence, these two variables, namely temperature and partial pressure, must be controlled most rigorously. In addition, since each chemical reaction proceeds at a certain rate, temperature control must be co-ordinated with this rate; thus, temperature control becomes time-temperature control.

F. R. Archibald(5) mentions catalysis as another variable in roasting. He states: "Little has been said about catalysis. It applies particularly to the oxidation of SO_2 and As_2O_3 and, except for attention to dust loads and point of dust return to the roaster, is not subject to control".

Thus, catalysis will affect reaction rates. Hence, catalysis must be considered as one of the factors in determining reaction rates.

Finally, to favour chemical reactions and both their rate and uniformity of rate, ores and concentrates must be ground sufficiently fine so as to obtain the optimum homogeneity for them, considering both the metallurgy and the economics of the process.

From the above discussion it can be seen that, fundamentally, roasting is the promotion of chemical reactions in their proper sequence.

In 1936, to check the laboratory roasting results obtained at Beattie Gold Mines, investigations were started at the Mines Branch on Beattie primary gold concentrate. This concentrate contained 4.95 per cent arsenic, 26.08 per cent sulphur, and 2.00 oz. gold per ton. Samples of this concentrate were roasted in an electrically heated muffle furnace with variable draft and *temperature control*. The charges, which were contained in vitreosil trays, were intermittently hand-rabbled.

R. J. Traill's investigations on Beattie concentrates at the Mines Branch established conclusively the necessity of controlling the rates of chemical reactions through the rigid temperature control: only by proper control of the rates of reactions was it possible to obtain the most desirable calcines.

The roasting procedure worked out at the Mines Branch(6) is described as follows.

A charge "was placed in the cold furnace. The temperature after 45 minutes heating was 360°C. (680° F.) with heavy arsenic fumes coming off. This temperature was maintained for about 45 minutes until the arsenic fumes diminished, and then raised gradually to 750°C. (1,382°F.). Total roasting time was 7 hours and the loss in weight 21 per cent".

After grinding portions of this calcine in laboratory ball-mills for 2 and 8 hours, the ground products were cyanided for 24 hours. The extractions of gold were 94.7 and 95.7 per cent respectively. The conclusions reached were:

"The high extractions appear to be the result of slow roasting with a maximum of fine grinding of the calcine, and cyanidation in a nearly neutral pulp".

The above roasting procedure consisted of three stages: pre-heating, arsenic distillation, and oxidation of sulphides. The amount of sulphur dioxide in the gases surrounding the charge in the third stage had to be strictly controlled, and any excess of it carefully scrubbed away. This procedure served as the basis for the design of the Beattie roasting plant.

Archibald(5) states some of the chemical reactions involved in roasting of Beattie concentrates to be as follows:

$$FeAsS=As+FeS \quad (1)$$
$$4FeAs_2=7As+Fe_4As \quad (2)$$
$$2FeAsS+2FeS_2=4FeS+As_2S_2 \quad (3)$$
$$FeAsS+3O_2=FeAsO_4+SO_2 \quad (4)$$
$$2As_2S_2+7O_2=As_4O_6+4SO_2 \quad (5)$$
$$4As+3O_2=As_4O_6 \quad (6)$$
$$4FeS+7O_2=2Fe_2O_3+4SO_2 \quad (7)$$
$$3FeS+5O_2=Fe_3O_4+3SO_2 \quad (8)$$
$$2SO_2+O_2 \rightarrow Catalyst \longrightarrow 2SO_3 \quad (9)$$
$$4FeS_2+11O_2=2Fe_2O_3+8SO_2 \quad (10)$$
$$FeS_2=FeS+S \quad (11)$$
$$FeS_2+O_2=FeS+SO_2 \quad (12)$$
$$2Fe_3O_4+SO_3=3Fe_2O_3+SO_2 \quad (13)$$
$$FeO+SO_3=FeSO_4 \quad (14)$$
$$2FeSO_4=Fe_2O_3+SO_2+SO_3 \quad (15)$$
$$4FeSO_4+O_2=2(Fe_2O_3.2SO_3) \quad (16)$$
$$CaCO_3+SO_3=CaSO_4+CO_2 \quad (17)$$
$$2FeAsO_4=Fe_2O_3+As_2O_3 \quad (>932^0F.,\ 500^0C.) \quad (18)$$
$$As_2O_5=As_2C_3+O_2 \quad (19)$$
$$FeS_2+3O_2=FeSO_4+SO_2 \quad (20)$$
$$4Fe_3O_4+O_2=6Fe_2O_3 \quad (21)$$
$$As_2O_3+xSO_3+nH_2O=As_2O_3.xSO_3.nH_2O \quad (22)$$

Archibald considers the first twelve reactions to be the most important and desirable reactions. The study of these twelve reactions establishes pretty well the mechanism of roasting of arsenical sulphide concentrates.

The first three reactions indicate that, in the early stages of roasting, arsenic elimination takes place under reducing conditions, which conditions are essential if the formation of arsenates is to be

prevented. These reactions are followed by three reactions which show the oxidation of arsenic to arsenious oxide, two of them also providing the sulphurous oxide required toward the maintaining of the necessary reducing atmosphere. The next six reactions (7 to 12) show the oxidation of pyrrhotite and pyrite with formation of more sulphurous oxide.

Using the equation

$$\log p = -\frac{A^1}{T} + B^1$$

where A^1 and B^1 are vapour pressure constants for arsenic, Archibald calculates the minimum distillation temperature of arsenic at atmospheric pressure to be around 605°C. (1,121°F.). He states that (with Beattie concentrates) with the evolution of sulphurous oxide from the charge, the volume of the arsenic vapour is reduced to about one-twentieth of the total gases in contact with the charge (= 38 mm.); hence the minimum temperature for distillation of arsenic will be 481°C. (898°F.).

For Beattie concentrates carrying 16 per cent sulphur and 2.4 per cent arsenic, the best temperature for arsenic elimination was found to be 900°F. (482°C.), the reduction of arsenic being effected by utilization of about one-third of the total sulphur content.

Using the formula

$$\log p = -\frac{A^1}{T} + B^1$$

for pyrite concentrates, vapour pressure constants for sulphur being given as $A^1 = 3{,}807$ and $B^1 = 8.26$(7), the minimum temperature for sulphur distillation at atmospheric pressure will be 435°C. (815°F.). This is approximately the initial temperature found to be the best for roasting of Lake Shore pyrite concentrates (26.6 per cent sulphur)(8), sulphur vapour being the only gas immediately in contact with the concentrates.

In both Beattie and Lake Shore roasts, the temperatures measured are those of the gases. It appears, therefore, that the above equation would give the approximate temperatures of gases required in the initial stages for optimum roasting of sulphide concentrates of Beattie and Lake Shore type.

After the elimination of the arsenic in Beattie roast and the elimination of the sulphur in Kalgoorlie and Lake Shore roasts, the roasts are completed under active oxidizing conditions, the approximate finishing temperatures being 1,300°F. (705°C.) for Beattie and 1,150°F. (620°C.) for Kalgoorlie and Lake Shore.

Table I shows the results of some of the roasting investigations carried out by the Mines Branch between 1939 and 1950 on concentrates from four different mines (9, 10, 11, 12, 13).

In all these tests the calcines were ground before cyanidation to approximately 95 per cent minus 325 mesh.

Comparing roasts Nos. 1 and 2, it would appear that much better roasting conditions were obtained in No. 2 than in No. 1. The writer ventures to postulate that this was due to higher initial temperature of roast No. 2, more nearly approaching the temperature given by the equation $\log p = -A^1/T + B^1$. With the higher temperature, the equilibrium partial pressure of arsenic vapour was greater which, with other factors (grind, thickness of bed, and scrubbing) being equal, promoted better reducing conditions for the arsenic elimination in roast No. 2. In No. 1, the initial temperature of 572°F. (300°C.) was maintained for 2½ hours until the arsenic elimination was practically complete. This is worth noting, since it shows that arsenic distillation from arsenopyrite can take place at a very appreciable rate at a relatively low temperature. However, it must be noted here that all temperatures in the Mines Branch roasting investigations were measured with a thermocouple placed about 2½ inches above the charge (about 5½ inches above the floor of the Globar electric furnace), so the temperatures indicated were actually lower than those of the roasts themselves.

It is difficult to compare roasts Nos. 3 and 4 on the basis of cyanidation results, since No. 3 calcine head is 4.7 times higher than that of No. 4. However, taking the formula $\log p = -A^1/T + B^1$ into consideration, it might again be postulated that, with an initial temperature higher than 752°F. (400°C.) in roast No. 4. a more porous calcine for cyanidation might have been obtained.

In roast No. 6, the high initial temperature of 815°C. (1,500°F.) was maintained for half an hour. After this time the charge was removed from the furnace and covered to exclude air. The furnace was cooled to 500°C. (932°F.) and the charge was replaced; the furnace door was left open to allow oxidation, the temperature raised to 650°C. (1,200°F.), and the roast was completed in 1¼ hours. Comparing calcine No. 5 with calcine No. 6, the cyanidation results indicate that, in both cases, good porous calcines were obtained. Evidently, in roast No. 6, during the high initial temperature stage, the arsenic and arsenious oxide fumes blanketed the charge completely, thus preventing the air from coming into contact with the charge. Hence, the conclusion appears to be inescapable that, by roasting in a completely reducing atmosphere, the harmful effect of the high initial temperature was practically eliminated. This would tend further to confirm Norwood's(4) original suggestion that the bad effect of high initial roasting temperature is due to flash-roasting. With higher initial roasting temperature, roast No. 6 was completed in much shorter time than No. 5.

Roasting investigations of outstanding interest were carried out at the Mines Branch on an auriferous arsenopyrite concentrate from the Nickel Plate mine, British Columbia, by Professor F. A. Forward in 1940 and were later continued by him in association with A. G. Lyle at the University of British Columbia(14). In 1949, Forward's roasting process for Nickel Plate concentrate was confirmed,

TABLE I.— RESULTS OF ROASTING TESTS

INE	ROAST NO.	ANAL. OF FLOT. CONC.			INITIAL TEMP		FINAL TEMP.		TIME OF ROAST	CALCINE HEAD, Au	CYANIDATION TAIL, Au	EXTRACTION,
		As %	S %	Ratio As:S	°C.	°F.	°C.	°F.	hr.	oz./ton	oz./ton	%
....	1	3.0	44.0	0.07	300	572	750	1392	5	1.48	0.07	95.3
....	2	2.52	42.3	0.06	400	752	700	1282	—	2.84	0.0475	98.3
....	3	9.94	15.1	0.66	480	896	550	1022	1½	4.40	0.33	92.5
C....	4	4.5	15.2	0.30	400	752	750	1382	3	0.935	0.0875	91.6
D....	5	26.30	19.10	1.38	480	896	680	1256	3½	4.03	0.17	95.8
D....	6	26.30	19.10	1.38	815	1500	650	1200	2¼	4.09	0.21	94.9

with minor variations, by the investigations of Dr. K. W. Downes, of the Mines Branch (15).

The Nickel Plate concentrate contained small amounts of cobalt and copper. The object of the roast was to obtain a calcine from which both the cobalt and copper could be leached out, after which the gold could be extracted by the cyanidation of the leach residue.

The analysis of the concentrates received at the Mines Branch indicated about 81 per cent arsenopyrite (FeAsS) and 5 per cent pyrrhotite (Fe_nS_{n+1}), the other metallic minerals present being pyrite (FeS_2), sphalerite (ZnS), chalcopyrite ($CuFeS_2$), löllingite-safflorite ($FeAs_2$. $CoAs_2$), cobaltite (CoAsS), and marcasite (FeS_2). Some gangue was also present. In addition, Warren and Thompson(14) state that there was also in Nickel Plate ore an undetermined grey mineral which might be a new bismuth telluride closely allied to joseite. The preliminary analysis of this new mineral suggested a formula Bi_5Te_2.

One of the concentrate samples sent to the Mines Branch analysed as follows:

Cobalt	0.50 per cent
Copper	0.48 "
Iron	31.62 "
Sulphur	18.17 "
Arsenic	37.60 "
Antimony	0.33 "
Gold	1.00 oz./ton

Forward's roasting process was investigated in an electrically heated muffle furnace, the concentrates being roasted in an open ceramic or silica dish.

This roasting process consisted of three stages:

(1) *Volatilization of Arsenic at* 750°-800°C. (1,382°-1,472°F.)

At these temperatures, arsenic distilled mostly as the element and partly as realgar and burned to arsenious oxide at a height of an inch or two above the charge, thus completely protecting the charge itself from oxidation. This condition continued, provided the temperature was maintained, until the arsenic in the charge was reduced to 0.7-0.5 per cent at the end of about 30 minutes. At this point, the arsenic distillation having practically ceased and there being not enough fumes to completely blanket the charge, the oxidation of the sulphides started taking place. This marked the end of the volatilization stage.

Forward and Lyle(14) say that not much sulphur is removed in the first stage. The low-arsenic calcine in their investigations contained 21 per cent sulphur and was a very soft, porous, and magnetic material, resembling pyrrhotite.

Though the arsenic volatilizes at temperatures below 1,382°F., lower temperatures cannot be used in stage (1) since then distillation of arsenic is insufficient to blanket the charge completely, which would lead to the formation of arsenates and copper and cobalt oxides. The arsenates would lock the gold, thus making it unrecoverable by cyanide solution, and the copper and cobalt oxides at temperatures over 1,050°F. would form insoluble ferrites with the iron oxides.

(2) *The Sulphating Roast*

Prior to this stage, the charge was withdrawn and cooled to about 500°C. (932°F.) and the muffle was cooled to about 450°C. (842° F.). The charge was then replaced in the furnace and roasted for about two hours at 842°F. to effect the oxidation of sulphide sulphur to sulphate sulphur, with the consequent formation of cobalt, copper, and iron sulphates.

Forward and Lyle(14) say: "The sulphating roast temperature can be readily controlled — very little air need be admitted — the less the better as long as there is sufficient to effect oxidation of the sulphur and iron. A high SO_2 content of the gases over the roasting material should be readily maintained. The extraordinary large surface area presented by the porous 'pyrrhotite' permits efficient use of oxygen in the air at the low temperatures conductive to SO_3 formation. The oxidation is highly exothermic, but the rate of reaction can be controlled by providing conditions — such as high SO_2 in the gases over the hearth — that tend to establish a desirable equilibrium and retard the burning rate".

Thus, it appears that in the sulphating roast the most important condition for the control of the rate of reaction is to maintain high equilibrium partial pressure of sulphur dioxide gas.

In the Mines Branch investigations it was found that, under the best conditions, the calcine obtained from the sulphating roast contained from 4 to 5 per cent sulphate sulphur and less than 0.1 per cent sulphide sulphur.

(3) *The Final Stage*

The final stage, recommended by Forward, consisted in heating the calcine to 625°C. (1,157°F.) in order to decompose the iron sulphates without decomposing copper and cobaltous sulphates. However, the investigations at the Mines Branch indicated that, to obtain maximum extractions of copper and cobalt by leaching the calcine, the finishing temperature should be as near the sulphating roast temperature as possible, no higher than is required to prevent unduly large amounts of iron from being dissolved in the leach solution, which may complicate the precipitation of cobalt and copper from the leach solution.

Table II shows the effect of final roasting temperature on the extraction of copper and cobalt(15).

TABLE II.— EFFECT OF FINAL ROASTING TEMPERATURE ON EXTRACTION OF COPPER AND COBALT

METAL	FINAL TEMP. 500°C. (932°F.)			FINAL TEMP. 600°C. (1.112°F)		
	WATER LEACH	ACID LEACH	TOTAL	WATER LEACH	ACID LEACH	TOTAL
Copper	76%	18%	94%	46%	30%	76%
Cobalt	89%	10%	99%	76%	17%	93%

Cyanidation of the acid leach residue, containing 1.38 oz. gold per ton, gave a tailing of 0.11 oz. gold per ton, or 95.3 per cent extraction.

The chemical reactions* which take place in various stages of roasting may now be suggested as follows:

STAGE (1)

$FeAsS = As + FeS$ ..... (1)
$CoAsS = As + CoS$ ..... (2)
$2FeAsS + 2FeS_2 = 4FeS + As_2S_2$ ..... (3)
$4FeAs_2 = 7As + Fe_4As$ (ref. 5) ..... (4)
$FeAs_2 + FeS_2 = 2As + 2FeS$ (ref. 16) ..... (5)
$CoAs_2 + FeS_2 = 2As + CoS + FeS$ ..... (6)
$2CoAs_2 = 3As + Co_2As$ ..... (7)
$4As + 3O_2 = As_4O_6$ ..... (8)
$2As_2S_2 + 7O_2 = As_4O_6 + 4SO_2$ ..... (9)
$4Fe_4As + 15O_2 = As_4O_6 + 8Fe_2O_3$ ..... (10)
$4Co_2As + 7O_2 = As_4O_6 + 8CoO$ ..... (11)
$CoO + Fe_2O_3 = Co(FeO_2)_2$ ..... (12)

Reaction (12) is an undesirable reaction, but judging by the results obtained at a final temperature of 500°C. (932°F.), shown in Table II, if there was any cobaltous ferrite formed in stage (1) it was in negligible quantity. This

* These are not all equally well established.

would indicate that reaction (7) for all practical purposes either did not take place or, if it did, very little of the Co_2As (oxidized to CoO) came into contact with Fe_2O_3.

STAGE (2), TEMPERATURE 450°C. (842°F.)

$$4FeS+7O_2=2Fe_2O_3+4SO_2 \quad (13)$$
$$3FeS+5O_2=FeO.Fe_2O_3+3SO_2 \quad (14)$$
$$FeS+2O_2=FeSO_4 \quad (15)$$
$$2CuFeS_2+O_2=Cu_2S+2FeS+SO_2 \quad (16)$$
$$4Cu_2+S4FeSO_4+11O_2=8CuSO_4+2Fe_2O_3* \quad (17)$$
$$CoS+2O_2=CoSO_4 \quad (18)$$
$$Cu_2S+2O_2=2CuO+SO_2 \quad (19)$$
$$2SO_2+O_2 \rightleftarrows 2SO_3 \quad (20)$$
$$2CuO+2SO_3=2CuSO_4 \quad (21)$$
$$6Fe_2O_3+2SO_2=4(FeO.Fe_2O_3)+2SO_3 \quad (22)$$
$$Fe_2O_3+3SO_3=Fe_2(SO_4)_3 \quad (23)$$
$$2Fe_2(SO_4)_3=2(Fe_2O_3.2SO_3)+2SO_3 \quad (24)$$

Care should be taken that the temperature of 842°F. be not exceeded since, according to C. J. Baroch(17): "Temperatures above 850°F. (455°C.) should be avoided during this stage (sulphating roast); otherwise, a true magnetite iron will be formed and this is difficult to re-oxidize to the insoluble form. The reaction here probably is:

$$FeS+10Fe_2O_3=7Fe_3O_4+SO_2 \quad (25$$

"This reaction explains why it is possible to raise the temperature above 850°F. (455°C.), after the first stage (elimination of sulphur) is complete, without the formation of magnetite".

However, according to Norwood (14) who, after the thermal decomposition of Kalgoorlie pyrite to pyrrhotite, carried out the reduction of calcine to magnetite at a temperature of 700°C. (1,292°F.), the magnetite, after all the sulphur was eliminated, was subsequently re-oxidized to ferric oxide. Norwood does not mention any difficulties in carrying out this re-oxidation at temperatures of from 500°C. (932°F.) to 800°C. (1,472°F.).

The mechanism of magnetite formation and its subsequent re-oxidation, confirmed by later experiments, was explained by Norwood(14) as follows:

"Just as, during the thermal decomposition of pyrite to pyrrhotite, the sulphur evolved partially or completely blankets the charge from the air, so, during the subsequent roasting, sulphur dioxide covers the charge and an important reaction is:

$$3FeS+2SO_2=Fe_3O_4+5S-17 \text{ cal.} \quad (26)$$

and the evolved sulphur burns above the charge to SO_2, thereby again more or less blanketing the charge from air, although this will be in intimate contact with the SO_2. Where the charge is subjected to a strong draught, as near the ports, the SO_2 will be scrubbed away and the normal air oxidation take place. While the charge contains a high proportion of sulphide, the reaction given above will be favoured. When, however, the oxidation of the roast is nearing completion, the sulphur-dioxide will no longer all be consumed by this reaction and the following reaction can take place:

$$3Fe_2O_3+SO_2=2Fe_3O_4+SO_3 \quad (27)$$

"Finally, when the roasting of the sulphides has been completed, the reaction proceeds:

$$4\,Fe_3O_4+O_2=6Fe_2O_3" \quad (28)$$

The above discussion of Norwood appears to contradict Baroch's statement. However, this contradiction can be explained by the fact that Baroch and Norwood roasted different sulphides from different localities. While the Bagdad ore at temperatures about 850°F. (455° C.) oxidized to "true iron magnetite", the Kalgoorlie ore might have oxidized at temperatures even as high as 1,292°F. (700°C.) to a magnetic oxide, expressed by formula $FeO.Fe_2O_3$. Tentatively, it might be suggested here that the mechanism of formation of magnetite in each particular case was due to the difference between the chemical structure of the Bagdad chalcopyrite and that of the Kalgoorlie pyrite. A suggestion was made at the Mines Branch that the nature of various compounds formed in roasting and their chemical structures might primarily depend on temperatures at which the original minerals had been deposited. This suggestion opens a promising avenue of research since, in the opinion of the writer, geothermometric studies of temperatures at which various minerals had been deposited might lead to explaining of natures of various compounds formed in roasting and defining their chemical structures.

From the reactions given for stage (2), it would appear that, by maintaining proper equilibrium conditions during the sulphating roast stage, such as high sulphur dioxide and sufficient oxygen in the gases, the dissociation of sulphates into oxides, sulphur dioxide, and oxygen can be prevented. However, E. A. Ashcroft (18) states:

"The ordinary rabbled furnace is inimical to the production of 'all water-soluble' copper sulphate by reason of the omnipresent reaction:

$$CuSO_4+CuS+O_2=2CuO+2SO_2 \quad (29)$$

"This reaction in well rabbled charges begins at almost as low a temperature as the oxidation of sulphur to form sulphates begins. Certainly it is in full swing at 400°C. (752°F.)".

Reaction (29) suggests the possibility of another reaction, namely:

$$2CuSO_4+2Cu_2S+3O_2=6CuO+4SO_2 \quad (30)$$

This appears to be confirmed by Baroch(17), who says: "Formation of sulphates can be regulated to a certain extent during this stage (sulphating roast) by controlling the temperature. From 700°F. to 720°F. (370° to 380°C.) seems to be the temperature of maximum sulphate formation, and temperatures above this give lesser quantities which vary almost proportionately with the increase in temperature. As high as 60 per cent of the total copper was present as copper sulfate, with temperatures of 700°F. (370°C.) during this stage, and only 15 per cent with temperatures of 850°F. (455°C.)".

Since, during the sulphating roast, the charges in the Mines Branch investigations were continuously rabbled, reaction (30) might have been taking place, and probably a similar reaction with respect to cobaltous sulphate. This is indicated by the results given in Table II, which show that quite appreciable amounts of copper and to a lesser degree of cobalt had to be extracted by acid leach.

According to Mellor(19) "cupric oxide may be employed as a contact catalyst for accelerating the union of SO_2 and O in the formation of SO_3". With the increase in the rate of formation of SO_3, it might be expected that the conversion of CuO and CoO to sulphates would also be accelerated. However, as long as there are Cu_2S and CoS in the charge, reaction (30) may be expected to be taking place whenever, at each turn of the rabbles, the sulphides will be coming into contact with the sulphates. Reaction (30) will cease taking place toward the end of stage (2), when all the sulphides are eliminated.

A factor which may also favour the formation of copper and cobaltous sulphates is the presence of iron sulphates. As stated by Ashcroft(18): "Iron is probably the only element, at least the only principal element, which by reason of the great heat of formation of its higher oxide, Fe_2O_3, will readily

* LIDDELL, D. M., *Handbook of Nonferrous Metallurgy*, 2nd Ed., McGraw-Hill Book Co., Inc., 1945, p. 279.

yield up its acid radical without decomposition, to any nearby basic material, such as oxidized copper or basic gangue constituents, and itself pass wholly to oxide in a sufficiently oxidizing atmosphere".

Hence the following reactions could be expected to take place:

$$4FeSO_4+4CuO+O_2 = 4CuSC_4+2Fe_2O_3 \quad (31)$$
$$Fe_2(SO_4)_3+3CuO = CuSO_4+Fe_2O_3 \quad (32)$$
$$4FeSO_4+4CoO+O_2 = 4CoSO_4+2Fe_2C_3 \quad (33)$$
$$Fe_2(SO_4)_3+3CoO = 3CoSO_4+Fe_2O_3, \quad (34)$$

Thus, it can be seen that the presence of iron sulphates may speed up the formation of copper and cobaltous sulphates.

Stage (3): Iron Sulphates Dissociation Stage

$$3FeSO_4 = FeO.Fe_2O_3+3SO_2+O_2 \quad (35)$$
$$4(FeO.Fe_2O_3)+O_2 = 6Fe_2O_3 \quad (36)$$
$$Fe_2(SO_4)_3 = Fe_2O_3+3SO_3 \rightleftarrows Fe_2O_3+3SO_2+1\tfrac{1}{2}O_2 \quad (37)$$
$$2FeSO_4 = Fe_2O_3+SO_2+SO_3 \quad (38)$$

Due to decomposition of ferric and ferrous sulphates, SO_3 is given out which would favour the conversion of the residual CuO and CoO to sulphates. Hence, if temperatures be kept sufficiently low during stage (3), it could be expected that most of the remaining oxides of copper and cobalt would be converted to sulphates.

Finally, during stage (3), the danger of formation of ferrites must not be overlooked. Ashcroft(18) states: "At any temperature above 550°C. (1,022°F), or thereabouts, formation of cupric ferrite, $CuO.Fe_2O_3$, begins when copper oxide and iron oxide are brought into juxtaposition and even at 700°C. (1,292°F) such formation is prohibitively rapid. It is, of course, well known that such ferrites are insoluble in any practical acid leaching solution".

Hence, at temperatures of 600°C. to 625°C. (1,112° to 1,157°F.) the following reaction should be anticipated:

$$CuO + Fe_2O_3 = CuO.Fe_2O_3 \quad (39)$$

Study of Table II appears to confirm this. At a temperature of 932°F. (500°C.) in the final stage of roasting, the total recoveries of copper and cobalt were 94 and 99 per cent respectively, whereas at 1,112°F. (600°C.) they were 76 and 93 per cent, *i.e.*, 18 per cent less copper and 6 per cent less cobalt were recovered when the final temperature of the roast exceeded the minimum temperature required for copper ferrite formation (1,022°F.) by 90°F. (50°C.).

The above discussion on the chemical reactions involved at certain temperatures in roasting of Nickel Plate concentrates establishes fairly well the temperatures at which various stages of roasting must be carried out for the best metallurgical results. It appears that the arsenic distillation must be carried out at 1,380° to 1,470°F., (750° to 800°C.), the sulphating roast at 700° to 720°F. (370° to 380°C.), and the final stage at 930°F. (500°C.).

These temperatures and their rigid control for each successive roasting stage form the basis for the design of the commercial roaster. A modified Edwards roaster, to roast Nickel Plate concentrates on a commercial scale, was proposed by Forward and Lyle. The writer is in agreement with the design of this roaster, which appears to incorporate all the necessary features requisite for successful roasting of Nickel Plate concentrates. The slight change in temperature for the second stage, suggested by him, and the lowering of the temperature for the third stage, based on Dr. Downes' work, do not interfere with this design except possibly for some minor mechanical modifications. However, the writer would like to qualify his statement by saying that any other type of roaster, besides Edwards, which meets the conditions of roast as discussed above and summarized in the previous paragraph, will be suitable for roasting of Nickel Plate concentrates.

Forward and Lyle state the reasons for the proposed design, and describe the roaster, as follows(14):

"During the course of the investigation much thought was given to this phase of the problem and, in consultation with manufacturers of roasting equipment, it was generally agreed that the double roast could be carried out in a single roaster of the 'Edwards' or 'flat-hearth' type. A similar operation has been performed in a plant treating tin sulphide ores using *three* stages of roasting. It is simply necessary to instal a baffle wall at a suitable distance from the feed end of the roaster, heat the chamber so provided to a temperature of 800°C., and withdraw the hot products of combustion which would carry the arsenic oxide (possibly also arsenic vapour) that distills so readily from arsenopyrite. We proposed having the next-hearth-in-line at a slightly lower level than the hearth of the first chamber, in order that the high-sulphur low-arsenic calcine could feed by gravity under the baffle wall. The second hearth would consist of water-cooled plates instead of bricks and would, in turn, be isolated from the third section (sulphating) by a baffle wall. The hearth of the third section was to be slightly lower than the second to permit cooled calcine to flow under the wall to this hearth. Air admitted to the third section was to be carefully controlled and, at the discharge end, some heat was to be applied to effect decomposition of the small amount of ferric sulphate that had formed.

"In brief, the roaster would be an Edwards-type machine divided into three sections by baffle walls, with the hearths of the second and third sections stepped below the level of the first and second respectively, the second hearth consisting of water-cooled plates instead of bricks".

Canadian Practices of Roasting of Auriferous Sulphide Concentrates

Summarizing the above discussion, study and evaluation of the following physico-chemical factors are fundamental in the control of roasting: temperature, partial pressures of gases, catalysis, and nature of roaster feed. Hence, to control roasting, these factors must be controlled and a roasting plant must be so designed as to make their control as nearly perfect as possible.

This chapter reviews Canadian practices of roasting of auriferous sulphide concentrates and analyses controls of physico-chemical factors, with a special emphasis on temperature controls.

Edwards Roaster Plants

The Edwards roaster, which was originally developed in Australia, is used in several Canadian gold mines to roast arsenical concentrates and, at Lake Shore, to roast a scavenger pyrite concentrate.

It is a single-hearth roaster, long and narrow, with rabbling being done generally by a double row of short rabble arms carrying rabble teeth or shoes, and revolving at speeds ranging from 1.5 to 6 r.p.m. (20,21). The rabble arms are attached to vertical spindles, which are rotated by bevel gears driven by a single-line shaft. Each arm rotates in a direction opposite to that of the adjacent arms, which makes the rabbling action of the Edwards roaster non-positive. This non-positive rabbling action, in the

words of Williamson(8), "causes considerable mixing back and forth in the roaster so that the cooling effect of incoming wet feed is exerted some five or six rabbles down the furnace, while hot charge on the point of decomposition is carried back toward the feed end. The frequent turnover ensures that material does not remain on top of the charge long enough to ignite during this preliminary stage of the roast. This back mixing effect serves throughout the roaster to prevent sudden rise in temperature which might lead to local overheating and a sintered calcine".

Other advantages claimed for the Edwards roaster by the same author are as follows:

(1) "Under properly controlled operation, the Edwards has been found to produce less dust than the other types (of rabble roasters). Unless arsenic recovery is necessary, the Edwards could be run without the expense of installing and operating a Cottrell or other dust collecting system.

(2) "In calcining self-roasting feeds it has been found possible to exercise a somewhat closer control on temperature in the Edwards.

(3) "The Edwards roaster is all on one floor, with all working parts visible from this single floor. A single operator can, therefore, give the roaster much closer attention than would be possible if it had to be served from several floors.

(4) "The usual repairs required by the Edwards, such as changing rabbles or rabble shoes, can be carried out from the outside of the furnace without emptying or cooling. The small size of rabble arms makes replacement rapid and simple".

At Lake Shore, special features were introduced into the standard design of the Edwards roaster to solve the problems inherent in the roasting of very fine pyrite concentrates, containing 26 to 27 per cent sulphur.

Special insulation was designed to control heat dissipation; a special air-cooled rabble for better control of roast temperatures was installed instead of a conventional water-cooled rabble; a pair of double-armed rabbles was installed at the feed end of the roaster; a brick chamber to obtain a reasonably gas-tight seal between the roaster and the gas piping, called the 'cathedral', was added to the feed end of the roaster; and the rabble shoe was modified to cut down as much as possible on the amount of build-up of the charge filling the space between the shoes.

The control of temperature in the Lake Shore Edwards roaster is described by Williamson as follows:

"(1) *Air Cooling of the Rabbles*: The use of air-cooled rabbles makes it possible for the operator to adjust furnace temperatures conveniently and accurately over a reasonably wide range, as the air-cooled rabble can be operated safely from cool condition up to a dull-red.

"In an air-cooled installation, the rate of heat removal depends on the velocity of air through the rabble. The greater the temperature gradient across the wall of the rabble, the greater will be the heat extraction.

"(2) *The Sulphur Grade of the Concentrate*: The concentrate is held at 26 to 27 per cent S. If the grade of concentrate in the storage tanks rises, it is brought down to 26 to 27 per cent by the addition of main plant cyanide residue.

"(3) *Moisture in the Filter Cake*: As an immediate first-aid measure, the roaster operator shuts off the slapper on the concentrate filter. This raises the moisture content of the roast feed by 25 per cent. This additional moisture produces a rapid drop of temperature at the feed end.

"(4) *Draft*: This control is not used. The efficient operation of the spray tower depends upon a constant supply of roaster gas containing 2.7 per cent (or slightly higher) of SO_2. Moreover, insofar as temperature control is based on the gas temperatures measured by thermocouples located at points along the roaster, any change in the draft would upset the correlation between charge and gas temperature".

Two major factors which contributed toward making the roasting of Lake Shore fine concentrates commercially possible were dust lubrication of the feed effected through the non-positive rabbling action of the Edwards roaster, and production of only very small amounts of dust, which obviated the necessity of installing a dust collecting system.

To obtain commercial cyanide extractions from calcines at Lake Shore, salt has to be added to the roaster charge. The addition of 30 lb. salt per ton of concentrates roasted shortened the total time of roasting from 10 to 20 per cent, imparted to the calcine a colour which had a rich purple-red shade, and raised the cyanide extraction from 70-80 per cent to 90 per cent.

The infrasizing of the calcine from salt roasts showed that this calcine was all minus 10 micron, and finer than the calcine from roasts without salt. According to Williamson, this "in reality probably indicated merely a more friable structure in the grains of hematite as a result of using salt".

The writer suggests that, in roasting pyrite with limited amounts of salt, ferric and ferrous chlorides are formed which are oxidized with the liberation of chlorine. The liberated chlorine attacks the sulphides, forming chlorides again which, in their turn, are also oxidized. Thus, a chain reaction is set in, the result of the oxidation of the ferric and ferrous chlorides being that a more porous calcine is produced in shorter time. Secondly, this calcine, when cyanided, yields more gold, originally occluded in pyrite either as colloidal gold or even in solid solution in pyrite, than would a calcine which was obtained by roasting without salt.

An Edwards roaster plant was also installed at MacLeod-Cockshutt to roast gold-bearing arsenopyrite concentrates. The MacLeod-Cockshutt roasting practice was described by R. C. Gegg(21) and differs somewhat from Lake Shore practice.

At MacLeod-Cockshutt, concentrates containing 3 to 5 per cent arsenic and 35 to 38 per cent sulphur are roasted in four Edwards duplex 60-spindle roasters, each roaster having a capacity of about 50 tons of concentrates per day. When installed, the rabble arms were water-cooled. However, due to the water-cooling causing a sharp drop in temperature gradient in the rabble arms, these rabble arms proved to be highly unsatisfactory mechanically, so eventually they were replaced by high-chrome low-nickel alloy (28 per cent Cr, 2 per cent Ni) rabble arms in the high-temperature zone of the three furnaces.

The primary advantage of these alloy rabble arms, Gegg states, "was that it made possible the discarding of the water-cooled arms, thus avoiding the radiation or heat extraction effect of the circulating water of the original arms. Other advantages appeared after the alloy arms have been in operation. No build-up of arsenic was noted between the arms and the lugs of the teeth, due to the fact that the arms were operating at a furnace temperature which is above the condensation point of the volatile compound which formerly deposited on the water-cooled arms. The teeth are loose on the arm and can be easily removed in the event of change of teeth or arm. Little or no

corrosion is noticed on the rabble arms after years of operation. Changes in these rabble arms are occasioned now only by breakages due to the presence of foreign material in the furnace (bricks, etc.) or to the spillage upon the arms of water from the water seals, causing cracking of the metal".

Regarding temperature control, Gegg says:

"Each furnace has ten thermocouples connected with a pyrometer at the operator's desk, and each is connected with a draft gauge. The first thermocouple is installed in the exit flue leading from the dust chamber to the fan. The others are distributed throughout the length of the furnace. Their locations and a typical series of temperature readings are shown on an accompanying graph. These readings are taken hourly and are representative of temperature conditions when roasting about 40 tons of concentrates per day".

The roast is started at about 540°F. and "the highest temperature in the furnace should not exceed 1,300°F. for best calcination and this is controlled by adjustment of air and draft conditions in the furnace. These must be varied to suit conditions caused by high sulphur content and fluctuating feed rates caused by variable amounts of sulphides in the flotation feed".

Finally, dust collecting is described as follows:

"At the feed end of each furnace is a dust chamber" which "acts as a settling chamber for the dust in the gases and also as a connection between the furnace and the two outlet gas flues. One outlet flue serves as the intake of a Sirocco fan; the other acts as a by-pass flue — it is directly connected to the outside balloon flue and can be opened in the event of a shut-down of the fan. A Sirocco fan handles the gas from the furnace, to overcome the resistance of the multiclones. The fan passes the gas to a multiclone dust collector.

"The dust from each multiclone is collected in a hopper which screw feeds the dust to a sump, where the dust is re-pulped with water. This pulp is returned to the 45-foot thickener where it is mixed with the incoming concentrates and eventually is fed back to the filters.

"The gases from the multiclones go to a common flue of balloon section, located outside the roaster building. This flue has a hemispherical top and a hopper bottom with outlet parts which can be opened to remove dust accumulation. The flue connects with a riser flue which enters the stack 40 feet from its base".

Two clean-out doors are provided at the base of the stack for the removal of the accumulated dust.

Wedge-Type Roaster Plants

The development of Wedge furnaces from the original MacDougall furnace was described by A. A. Dwight(22), and the operation of a Wedge roasting plant at Beattie was described by Archibald in his paper on Beattie roasting practice(5).

Three 25-foot-diameter Wedge-type furnaces to roast gold-bearing arsenical concentrates were installed at Beattie. Each roaster has 13 hearths and a dryer hearth, the approximate hearth area being 4,500 sq. ft. During installation, particular attention was paid to the insulation, more than usual insulation being provided for each roaster. The roaster shafts and rabble arms are air-cooled. The shaft carries two rabble arms per hearth and makes approximately one revolution per minute. The concentrates are dried in a Ruggles-Coles dryer to 8 to 10 per cent moisture, then conveyed to roaster feed bins from which they are charged to the upper hearth. The total time of roast, which is autogenous, is 4½ hours. Two roasters are ample to handle the present feed tonnage.

Archibald describes the dust collecting as follows:

"Each roaster was provided with gas bleeder flues from hearths 7 and 11, and two uptake flues from hearth 1. These flues joined to enter a multiclone dust collector at the top of each roaster, and these collectors, in turn, each discharged through a fan to a common balloon flue. The flues were all brick lined.

"The balloon flue discharged into a humidifying chamber, which in turn discharged into the hot Cottrell unit. The discharge flue from the latter led to a fan, the inlet of which was provided with an automatically controlled louvre damper and flue to admit cooling air from outside the plant. The hot Cottrell gases and atmospheric air were mixed by the fan and the mixture so regulated that the temperature was maintained at 250°F. (121°C.) in order to ensure efficient condensation and precipitation of arsenious oxide in the cold Cottrell unit following. The hot and cold Cottrell units each consisted of two sections, each section carrying 14 rod curtains (13 ducts), 12 feet long.

"The cold Cottrell discharged through a flue to the 400-foot brick-lined, reinforced concrete stack, 18½ feet diameter at the bottom tapering to 7 feet diameter at the top, inside dimensions.

"Dust from the multiclone collectors returned to the 6th hearth by gravity. The dust from the hot Cottrell was collected from the Cottrell hoppers by drag conveyors and elevated to return through a splitter to the sixth hearth of the two roasters used".

During a period which was selected as being typical, "the average daily dust tonnages were 15.38 tons from the multiclones (total for both roaster units operating), and 15.4 tons from the hot Cottrell. This represents a total dust load of 13.4 per cent of the dry feed".

The first requisite in the control of Beattie roast has been maintaining a constant sulphur content in concentrates, which has meant seasonal adjustments in flotation as the sulphur content in the ore has kept changing. This constant sulphur content of the roaster feed must be kept well within 15 to 16 per cent, since, as the author says: "Not only does the sulphur from pyrite serve as a source of heat, and as a reducing agent, but the volume of gas generated from it within the charge serves to sweep out the volatile arsenic compounds somewhat in the manner of zinc distillation in carbon monoxide from zinc oxide-carbon mixtures".

Concerning other controls, Archibald says:

"Temperature, draft, and air inlet control are interdependent and these were the chief manipulative controls required of the operators. The draft balance varied seasonally and with the tonnage, but for average conditions the static draft graded for 0.6 inches of water at the stack to 0.25 inches or lower at the roaster flue outlets. The relative gas volumes drawn through the uptakes on hearth 1 and the bleeders from hearths 7 and 11 were regulated by dampers to keep the roaster temperatures in balance. The uptakes were normally wide open and 60 to 70 per cent of the total gas volume was drawn through them. Number 4 hearth was considered the key hearth in controlling the temperatures and it was maintained as nearly constant as possible for a given tonnage rate. If a roaster showed a tendency to heat-up at the top, the bleeder on hearth 11 was opened wider and, if necessary to re-establish a balance, the up-draft in it was boost-

ed temporarily by means of an air jet. Hearth 10 was normally the hottest hearth (gas temperature about 1,300°F. (705°C.), and the charge cooled to 600° to 800°F. (315° to 427°C.) on hearth 13 before discharging to the quench tank.

"Air was admitted to the roasters by opening sight flaps and doors as required to maintain the temperatures and complete the roast".

In conclusion, Archibald makes the following remarks:

"The picture is far short of perfection, from both theoretical and practical viewpoints, and there remain unanswered problems in both directions. Roasting tests in the Beattie laboratory numbered in the hundreds, but more fundamental studies in physical and structural chemistry are needed than are possible in even well equipped mining company laboratories".

The writer would like to enlarge on these remarks by saying that the art of roasting has been far from perfection because it has been based too much on empiricism instead of on a concerted study of fundamentals which involve not only the physico-chemical factors but also microstructures of minerals which are a function of the depositional environment.

FluoSolids Roaster Plants

Recently, a new technique, originally developed in the petroleum industry, in which concentrates are fluidized, has been applied to roasting of sulphide concentrates. The application of fluidization in roasting, now known as FluoSolids roasting, in the fields other than petroleum refining, appears to hold great promise for better control of temperature than has been possible so far with rabble-type roasters.

The first FluoSolids roaster plant in Canada went into operation at Cochenour Willans early in 1947. O. J. Matthews(23) summarizes the implications of fluidization in roasting as follows:

"The term 'fluidization' denotes the fundamental principle of carrying out gas-solid reaction in a dense suspension of solids, maintained in a turbulent mass by the upward flow of gases that effect the reaction. This mass assumes a fluid level and acts as a fluid. The upwardly-moving gas stream imarts to it a turbulence resembling that of a boiling liquid.

"The outstanding advantage of this principle, as embodied in the fluo-solids system, is the close degree of temperature control realized and the uniformity of the temperature condition that may be readily maintained throughout the fluidized bed. Under normal conditions, there is no appreciable temperature difference between different parts of the bed, nor is there any appreciable temperature difference between the gaseous phase and the solid phase".

From the above, it is realized that, as T. B. Counselman(24) says, the fluidized mass "will have a uniform consistency as to particle size, chemical composition, temperature, etc. The fluidized mass will exert hydrostatic head, and pressure taps will show differential pressures between points at various depths in the bed. The top of the fluidized mass although irregular, like the surface of a boiling liquid, has a sharp line of demarcation from the gas above. Most important, every solid particle is individually surrounded by a film of gas or air".

Advantages and disadvantages of fluo-solids roasting are described by Counselman in his article(24). Matthews states that: "The principal limitation of the process is that there must be no softening or fusion of the material undergoing treatment. Agglomerates or nodules resulting from fusion will not fluidize but will be deposited at the bottom of the fluid bed. This deposition might well lead to a condition which would result in the whole bed becoming static and de-fluidized".

Flotation concentrates at Cochenour Willans contain about 5.3 per cent arsenic, 16.6 per cent sulphur, and 6 oz. gold per ton. These concentrates are dewatered in a thickener and filter in series, conveyed at 12 per cent moisture to a bin, and thence are fed into the roaster or reactor by a Coghill-type feeder. This feeder consists of a vibrating hopper and a long, ribbon-type screw conveyor, which is also designed to act as a seal to prevent gas escaping from the roaster at this point.

The reactor or roaster consists of a cylindrical shell built of ½-in. steel plate, 18 feet high and 6 feet 8 inches inside diameter. The shell is lined by 9-in.-thick firebrick with a 3-in. backing of insulating brick and has a domed top lined with a castable refractory cement. The hot-air ducts and cyclone dust collectors are also lined with refractory cement. The roast is autogenous.

Describing handling of dust, Matthews says: "There are two outlets in the dome top of the reactor, one, 14 in. dia., being a gas discharge to the cyclone dust collectors, and the other, 8 in. dia., an auxiliary stack opening which is used only when the reactor is being preheated prior to the commencement of a campaign. Both are lined with refractory material. The main 14-in. gas line leads to two cyclone dust collectors arranged in series. The stack is 14 in. dia., built of ⅛-in. steel plate, and is 125 feet high".

The capacity of the reactor at Cochenour Willans is 15 tons per twenty-four hours. However, since only about 8 tons of concentrate are roasted daily, the reactor is operated intermittently.

Matthews says that: "It can be shut down and started up without difficulty and without any apparent disturbance in metallurgy or in extraction. The bed holds its temperature for a considerable time, the reactor being shut down on occasion for a period of 16 hours, and started up at operating temperature. In shutting down, all that is required is to cut off the feed and air".

When carrying out fluo-solids roasting tests on Cochenour Willans concentrates, it was found that excess of air, just as insufficiency of air, had an adverse effect on the production of porous calcine. The best roasts were obtained when, roasting the concentrates to a 'dead' roast, there was only a trace of oxygen in the exit gas.

This necessitated a very close control of the air input, which is done by adjusting the air valves admitting the air to the reactor. This meant analysing the stack gas for oxygen. However, at Cochenour Willans the gas analysis was substituted by visual control. This consists in watching the colouring of the stack plume. When the stack plume is white bordering on yellow, it indicates that there is just sufficient oxygen for producing the best roasting conditions.

Such a close regulation of air input means that the partial pressure of oxygen inside the roaster is of utmost importance for obtaining the proper thermodynamic conditions which would promote the desirable reactions, prevent the undesirable ones, and produce a calcine of the required porous structure.

It is evident from the above that the points where the concentrates are fed and calcines discharged must be sealed as effectively as possible against the escape of gases. At the feed end, as was mentioned before, this is done by the special design of the screw conveyor, and at the discharge end it is done by water seals at the quench tanks. The stack

is the only outlet for the reactor gases.

Regarding the temperature control, Matthews says: "Temperature control is very simple, consisting of injecting water into or onto the fluid bed at one point. The very high rate of heat transfer in a fluid bed stabilizes temperatures throughout, in spite of the heat extracted at the point of water introduction. Experience has shown that such temperature control at about 1,100° F. (593°C.) has resulted in reducing the gold content of calcine tailings from a high of 1.15 oz. to a low of 0.38 oz., and to a recent average of about 0.5 oz. from a calcine head of 6.3 oz.

"Thermocouples are connected at the lower and upper parts of the bed, at the top of the furnace in the gas chamber, and at the base of the stack. These record temperatures automatically every 4.6 minutes. As the bed under normal conditions is always in a turbulent fluid state, the thermocouples at the lower and upper parts of the bed should record approximately the same temperature".

Recently, FluoSolids roaster plants for roasting auriferous arsenical sulphide concentrates were installed at Campbell Red Lake and New Dickenson mines and are in operation now. Two more are being installed, one at Giant Yellowknife and one at Negus.

Conclusions

The understanding of physico-chemical factors is the essential requisite for good roasting and is fundamental in the design of roaster plants. Without this understanding, the pilot stage becomes very much a hit-and-miss affair and is unnecessarily prolonged.

The physico-chemical nature of roaster feed, which determines the ratio of reactants, temperature, partial pressures of gases, and, finally, catalysis, controls chemical reactions inside the roaster. For instance, as the percentage of arsenic in the concentrates increases, it appears that higher initial temperatures are required for successful roasting. This indicates that the initial temperature is determined by the partial pressure of the arsenic vapour, which pressure in its turn is determined by the arsenic-sulphur ratio.

For arsenopyrite concentrates with arsenic content ranging from 2.5 to 10 per cent, initial temperatures of roast in presence of sufficient sulphur are fairly well determined by the formula

$$\log p = -\frac{A}{T} + B$$

However, when roasting concentrates with high arsenic content, as for example Snow Lake or Nickel Plate concentrates, the initial temperatures required are higher than given by this formula: they must be sufficiently high to cause rapid volatilization of arsenic in order to blanket completely the charge and exclude the air from contact with the charge.

The presence of a catalyst increases the rate of reaction or reactions; thus the catalysis is another factor in determining the capacity and efficiency of the roaster.

In studying the Forward process of roasting the complex Nickel Plate concentrate, several reactants have to be considered since the ratios of these reactants are one of the chief factors determining the roasting procedure.

While the addition of salt to Lake Shore concentrates in roasting improved the porosity of Lake Shore calcines, salt additions to similar concentrates from other mines failed to show any change in calcines. Thus, while in the case of Lake Shore it was necessary to add salt to obtain good porous calcine, no salt was necessary in other cases. This would indicate that studies of structural chemistry of the material to be roasted are pertinent in roasting investigations, and it is further suggested that these studies might be based on geothermometric determinations.

Comparing roasting by rabble-type roasters with that by FluoSolids reactor, it is indicated that the fluidization technique produces a system in which the thermal equilibrium and the mechanical equilibrium are easily attained, *i.e.*, certain 'steady state' conditions are produced which allow chemical reactions to proceed in certain directions at certain rates until chemical equilibrium is established. In rabble-type roasters, the thermal and mechanical equilibria are liable to be continually upset due to rabbling, hence special mechanical means for temperature control and intermediate scrubbing of gases along the roaster must be introduced. Naturally, to make these mechanical controls foolproof is practically impossible. However, it must be noted that while, in Edwards and Wedge-type roasters, temperatures can be varied from one end to another, as already described, this cannot be done in a FluoSolids reactor. Hence, it is suggested that in multi-stage roasting when using the fluidization method, either the reactor should be divided into compartments or as many reactors should be used as there are stages in roasting.

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DISCUSSION

PROFESSOR F. A. FORWARD (Department of Mining and Metallurgy, University of British Columbia): The author is to be complimented on the work he has done in preparing this interesting paper on roasting and in bringing the topic again before the Institute.

A paper by Carter and Samis on *The Influence of Roasting Temperature upon Gold Extraction by Cyanidation from Refractory Gold Ores* which is to be published shortly in the Institute Bulletin*, describes some of the aspects of roasting which have been studied within recent years at the University of British Columbia. Carter and Samis believe that, in general, the reason for low recovery of gold when cyaniding a roasted concentrate can be attributed to particle fusion with consequent occlusion of the finer gold particles. Impurities such as lead and antimony tend to lower the fusion temperature of the particles and so increase the possibility of occlusion. Their work shows also that the temperature of the individual particles in a hearth roaster is probably 400°C. higher than the temperature measured by a pyrometer in the roasting bed. They suggest that best recoveries can be obtained by using low roasting temperatures or by controlling the temperature of the individual particles. This latter can probably be best accomplished in a roaster of the fluidized bed type.

MR. F. E. LATHE (National Research Council): It is clear from both the paper and discussion that, when calcines are to be subjected to chemical treatment, the particles being roasted must not be subjected to very high temperatures. Anyone can see by looking into a multiple-hearth furnace that, as the particles fall from one hearth to another, they are rapidly oxidized and are subjected to a temperature which may be several hundred degrees higher than that recorded by pyrometers located over the hearth. The Edwards roaster has partly overcome this difficulty, and the Dorrco FluoSolids reactor seems to have made another important advance. I am familiar with the close temperature control effected in FluoSolids reactors, both within and above the charge, but wonder whether individual particles of, say, minus 325-mesh material, which may at the bottom of the charge come into contact with air containing the normal percentage of oxygen, will be even momentarily overheated. Might it be necessary, for perfect temperature control, to re-circulate part of the gases?

MR. W. T. MARSTON (The Dorr Company, Inc.): Although thermocouples have been installed in almost every position and in great quantity in the fluid bed zone of FluoSolids reactors, we have never been able to detect any general zone or area where higher or lower temperatures beyond the range of plus or minus 20°F. existed.

Professor Forward has mentioned that the investigations of Carter and Samis have shown that partial fusion of particles, with consequent occlusion of the gold they carry, is responsible for low recoveries of gold when roasted concentrates are cyanided, and he suggested that this might apply also to FluoSolids calcines. As a result, however, of recent re-checking of our test work, and on the basis of our previous microscopic studies, it seems safe to state that such sintered particles do not exist in the calcines produced by a FluoSolids furnace, always provided that the reaction is carried out below the sintering or softening point. Further, in some of our early test work, comparative roasts were made using air as the fluidizing agent and an air-SO_2 mixture (6.5% SO_2, 11.0% O_2, 82.5% N_2). These comparative tests gave essentially identical gold extraction and therefore it was concluded that re-cycling of stack gases would not be necessary.

While we are in agreement with the discussion on the theoretical effect of oxygen upon the roasts, because of the violent agitation in the fluid bed and the instantaneous dilution of the feed with roasted product, it is difficult to follow the actual progress of roasting in a FluoSolids unit. However, we visualize that in FluoSolids operation each particle of feed becomes surrounded with hundreds of fully roasted calcine particles.

Also, since the mixing of the solids and gases is lateral, as well as vertical, the fresh feed particle will come in contact with gases containing various mixtures of SO_2 and O_2. Thus, the feed particles become surrounded with a film of SO_2-bearing gas almost immediately upon introduction into the reactor. This protective gas film, plus the mobility of the particles and the high heat transfer rates in fluidized operation, prevents flash roasting and sintering of the particles.

While the temperature of the individual particle must rise above the temperature of the fluidized mass during the oxidation of the particle, the temperature differential is very low because of the resistance offered by the gas film to flow of oxygen into the particle and because of the very efficient heat exchange between the particle and surrounding mass. The uniform temperatures measured throughout the fluidized bed are due to the violent agitation of the mass. It is estimated that 50 to 60 per cent of the total power requirements are utilized for agitation of the bed.

To digress from the actual oral discussion following the paper, we would like to comment on Mr. Djingheuzian's deductions and conclusions regarding the required mechanism for successfully roasting arsenical concentrates. These were particularly gratifying to us, since we had arrived at the same general conclusions. Negus Mines has recently installed a Dorrco FluoSolids system, in which the two-stage arrangement is utilized. The first stage eliminates arsenic under reducing conditions, such that arsenic sulphides are present in the gas stream and at a temperature of approximately 600°C. The partially treated calcine proceeds to the second stage, where the calcine is finished under strong oxidizing conditions at a temperature of approximately 650°C. The unit has not been in operation long enough to assess the results completely, but

* See March 1952 BULLETIN, pp. 160-166 (*Transactions*, Vol. LV, 1952, pp. 130-136).

we have every confidence that it will be completely successful. The FluoSolids reactor at Negus is divided vertically into two halves, so that one half functions as a primary stage and the other as the secondary stage.

You will also be interested to know that our standard method of feeding concentrates into a FluoSolids reactor is to meter them in with a constant displacement pump at 75 to 80 per cent solids. This has been applied to all the gold concentrate roasters in operation at the present time. You may be surprised to learn that even Cochenour Willans, with their low heat value concentrates, are successfully utilizing this method and maintaining their reactor at 1,100° to 1,200°F. This is peculiarly indicative of the efficiency of the FluoSolids furnace since, as stated by Mr. Djingheuzian, the sulphur content of Cochenour Willans concentrate was, in 1949, in the neighbourhood of 16.6 per cent, and the arsenic content about 5.3 per cent. Since that time, the sulphur content has decreased and the arsenic content increased, bringing them much closer to the break-even point on heat balance.

Mr. P. J. Ensio (Plant Manager, Mount Royal Metal Company, Limited, Montreal): Roasting is a chemical reaction between a solid sulphide-mineral phase and a gas phase, containing oxygen. Therefore, we must consider what takes place during the reaction in each of the two phases, as well as in the interphase. The relatively meager amount of research work in this field indicates that much about roasting can be learned through this approach.

We have recently carried out a series of tests in which fine particles of homogeneous pentlandite mineral were partially oxidized under controlled conditions. Separation of the outer oxide layer from the remaining sulphide in each particle showed a remarkably higher ratio of iron to nickel in the oxide phase than in the sulphide phase.

We know that most of the sulphide minerals have a high degree of disorder which greatly facilitates interstitial diffusion. Cat-ion diffusion rates, in fact, have been found to be, at moderate temperatures, of the same order of magnitude as those in the aqueous solutions. We must, therefore, look upon sulphide minerals as very mobile and reactive media. In the complex sulphide minerals, differences in diffusion rates of the cat-ions, as well as differences in the free energies of oxide formation of the metals in question, will have a controlling effect on the roasting process of each sulphide particle.

Mr. A. L. Blomfield (President, Lake Shore Mines): The Edwards roaster was developed in the State of Victoria, Australia, to roast gold concentrates for further treatment. These roasters were the single-rabble tilting type. In competition were several other types.

It was introduced about 1897 at Kalgoorlie and a dozen other camps in Western Australia. Preceding them and still running in 1901 were several other types: the 'Brown straight line', the Holthof, the Holthof-Wethey, and later the Morton, which is a double-deck Edwards. About 1904 came the Edwards Duplex.

In Colorado during this time, the Holthof and Holthof-Wethey and a variety of single and double hearth circular types were in general use. In 1905, the Edwards tilt-type was introduced in Cripple Creek; next, four Edwards Duplex machines were put in alongside four Holthof-Wetheys at the Golden Cycle property.

In all these districts, all types disappeared except the Edwards. It looks, on first view, the most complicated. In use, it proved very simple and the best for both costs and flexibility. It was generally considered the best of its group. When air cooling of the rabbles was added (design of D. L. Cramp, of Lake Shore), its work was practically revolutionized. The rabbles were identically the same as those in Kalgoorlie in 1906, or earlier, and the air joint was put on without stopping the roast.

Dust. — The main places to control dusting are:

(1) At feed — wet or into the hearth level.

(2) At point of maximum rabble speed, past a certain rather low speed, where the dust rises; the rabble must be watched to determine this point. The small rabble has a minimum speed for any frequency at ore turn over.

(3) Where the dry product is dropped from one deck to another against the draft.

(4) Where exploding wet feed is doused into hot part of roast.

(5) Where the finished product is doused while still not sufficiently cooled.

Never in my experience has the Cottrell Company advised installation of dust collectors after making tests on dust loss on any Edwards roaster my staff was running. Carelessness in operating the machines is another story.

The Edwards furnace is, I believe, the most flexible and adaptable milling machine on Lake Shore ground. It not only is easily regulated for heat at all points in the roast, it keeps at the same time a steady-type gas flow to the conditioning plant for tailing float. This conditioning is done at a cost of less than one cent per ton and, incidentally, saves more than that in the two or three cents per ton gold dissolved and otherwise lost in the tailing, as well as a high cost of reconditioning by purchased chemicals.

Residue. — Some remarkable residue results have been made and are being made with this roaster but, under poor operation, residue losses are about doubled. As we all know, some of the best machines in gold milling have given way to simpler machines, with poorer results. The management is still quite as important as the machine. A roaster is not much concerned with the free cyaniding gold in its feed. The residue must be stated in terms of the extra solubility obtained.

Let

A = free gold in the feed
B = the gold to be freed by roast
$A+B$ = total head value
R = residue

The extraction per cent due to roasting is *not*, as often given,

$$\frac{(A+B-R)\times 100}{A+B}$$

but is

$$\frac{(B-R)\times 100}{B}$$

e.g., if

$A=70.00,\ B=14.00,\ R=2.00,$ *then*

$$\frac{(A+B-R)\times 100}{A+B}=97.6\%$$

while true extraction is

$$\frac{(B-R)\times 100}{B}=85.7\%$$

Until this is recognized as the only accurate figure, we cannot get at comparative roasting residue efficiency, nor evaluate two types of roaster from the residue angle, which is often the most important point at issue.

The Salt Roast at Lake Shore.— The assayed mesh analysis of the Lake Shore calcine residue in 1933 was, as you will remember, a reverse curve — highest at both ends of the analysis. That, in my exper-

ience, is almost a certain indication of chemical failure, and on that conclusion I suggested trying the salt roast. It did, I believe, break down the mesh more than did the simple roast. Also, it did make its reduction in the finest part of the product.

Norwood in Kalgoorlie has, I think, blamed the high residue on lead-gold fusion and has credited the improved result to removal of the lead by the salt. There is enough lead sulphide present, and enough inclusion of gold in the lead telluride, to add weight to this suggestion. Williamson's* suggestion is also in order.

Dust Lubrication on 'wet-sticky' or 'heat-sticky' product. — The feed at Lake Shore is very sticky as fed. Only the return of a part of the finished material makes the furnace handle it. This might be the key to continuous operation of oil shale distillation. It was the handling of this sticky material that beat us twenty-seven years ago in Denver, Colorado,

The work we did in 1936 on handling such a product gave the most surprising reactions of any thing so far done at Lake Shore.

Others who contributed discussion on the paper included Mr. B. S. Crocker, Lake Shore Mines, and Dr. K. W. Downes, Mines Branch Ottawa.

Mr. Djingheuzian: I wish to express my deep appreciation to all who contributed to the discussion and thus enhanced the value of the paper by their thoughtful comments and criticisms.

Professor F. A. Forward's summary of the highlights of roasting investigations carried out on Con concentrates by Dr. Samis and Mr. Carter at the University of British Columbia makes a notable addition to the paper. The most important comment in this contribution is the one concerning particle fusion. Norwood(4), who experimented with Edwards roasters, terms this particle fusion 'flash' roasting, which is the bane of the operator roasting auriferous sulphide concentrates. As Mr. F. E. Lathe pointed out, 'flash' roasting might be well observed in a multi-hearth furnace where particles fall from one hearth to another, and he suggests the possibility of 'flash' roasting of minus 325 mesh material at the bottom of the charge in a FluoSolids roaster. This is also suggested by Matthews(23) in his paper when he says: "The principal limitation of the process is that there must be no softening or fusion of the material undergoing treatment".

While the mechanism of the operation of the FluoSolids roaster is so designed as to maintain proper thermal and mechanical equilibria to enable chemical reactions to proceed in the right direction, Mr. Lathe's remark points out the possibility of these equilibria being destroyed in certain locations of the roaster and under certain conditions. I believe that his suggestion of recirculating part of the gases for *perfect* (italics are mine) temperature control merits full consideration. This appears to be confirmed by Norwood(4) who says: "When the roast is made in sulphur dioxide, the oxidation of pyrite and pyrrhotite becomes endothermic, and the temperature of the grains cannot rise above that of the furnace, with the result that the harmful effect of the high temperatures is very greatly reduced".

Presumably, 'flash' roasting also takes place in FluoSolids roasters when there is an excess of oxygen.

Mr. B. S. Crocker's (Lake Shore Mines, Ltd.) very illuminating remarks that the non-positive action of the rabbles of the Edwards roaster does practically eliminate 'flash' roasting explains why this roaster has always been favoured by so many operators.

Professor Forward also notes that the work of Samis and Carter showed that the temperature of individual particles in a hearth roaster is probably 400°C. higher than the temperature measured by a pyrometer in the roasting bed. I have to go back to the remarkable work done by Norwood(4). He says: "The roast is far more quiescent under sulphur dioxide than under air, and it was, therefore, felt that although the temperature of the roast has always been measured by a bare metal thermocouple, the surface temperature of the charge in the latter case would be found to be much higher due to the highly exothermic reaction causing 'flash' roasting".

It appears to me that a rather searching discussion confirms the main thesis of my paper, namely, the absolute necessity of maintaining thermal and mechanical equilibria during the roast. Dr. K. W. Downes, in his remarks, said that it seemed to him that I endeavoured to show that principles of thermodynamics could be applied to roasting. And so it was, since, during the preparation of the paper, I had always thought of the mechanism of roasting in terms of equilibrium conditions which should be maintained in the roasters.

Mr. A. L. Blomfield's very able and succinct summary of the essence of the Edwards roaster is a remarkable addition to the paper by an operator of long standing. His statement that the extraction due to roasting is given by the formula

$$\frac{(B-R) \times 100}{B}$$

is fundamental. I fully agree with him that this formula must be recognized by all operators, since it gives the only accurate figure and tells the true story of roasting.

Mr. Blomfield's other statement that "the management is still quite as important as the machine" makes a fitting conclusion to my paper.

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* See chapter "The Use of Salt in the Roast," reference 8, p. 180.

DISCUSSION

ON

Metallurgical Practice at Golden Manitou

By D. A. LIVINGTONE

Transactions, Volume LV, 1952, pp. 241)

Mr. F. Clyde Lendrum (Ascot Metals Corporation, Limited): In the paper it is stated that when the reducing power of the cyanide circuit increased, zinc recovery fell off. Was this due to the zinc minerals being 'slower floating' or was it more difficult to depress the pyrite and, therefore, recovery was sacrificed to secure grade?

Mr. Livingstone: I would say that the higher reducing power in the cyanide circuit caused the sphalerite to be slower floating in the subsequent zinc flotation step. There was very little investigative work done along these lines owing to the brief period that this flow-sheet was in use. It has been theorized that, since such reducing agents as sulphur dioxide and sodium sulphite are depressants for zinc, reducing agents carried through from cyanidation may likewise have had an adverse effect on zinc flotation. The higher reducing power did not appear to hinder pyrite depression and, therefore, the zinc concentrate grade was not noticeably affected.

Mr. R. C. C. Palmer (Waite Amulet Mines, Limited): In connection with one of the flow-sheets, it is stated that depression of zinc was not satisfactory when sodium sulphite was used as a depressant. We are using sodium sulphite for zinc depression but I do not know whether we are depressing zinc or whether copper floatability is being improved to the point where it appears that depression of zinc is taking place.

Mr. Livingstone: Your remarks concerning sodium sulphite as a zinc depressant are of interest because, in our de-leading operation, where the aim is to depress zinc and float lead and silver minerals, we used sodium sulphite for a time, not because we could attribute to it any noticeable zinc depressing qualities, but because it seemed, on occasions, to assist in the floating of the copper-silver tennantite-tetrahedrite minerals. This, in a way, lines up with your thinking on the subject, only you are speaking in terms of chalcopyrite instead of the silver minerals which we have

Mr. M. J. Bennett (Quemont Mining Corporation, Limited): In the course of your work, have you found any reagent particularly suitable for the flotation of silver or silver minerals?

Mr. Livingstone: We tried a great many promoters for this purpose but, with the exception of *Reagent 208,* which showed some slight benefit in floating native silver in the primary flotation circuit, we have no marked preference for any one promoter specifically for floating silver minerals.

Mr. C. G. McLaughlan (Noranda Mines, Limited): Do you find that the flotation of zinc tends to lag behind the pyrite in your laboratory tests?

Mr. Livingstone: Assuming that you are referring to primary flotation, the answer is *No.* Using the floatability of pyrite as a basis for comparison, the flotation of zinc is markedly slower in laboratory tests than in the plant. As the flotation period increases, the difference diminishes, and, at the end of a period equivalent to the contact time in the plant, zinc recoveries become practically identical. This behaviour is not invariable but represents the usual trend.

Mr. McLachlan: I am interested in your answer because I have come across several ores in which the zinc floats more slowly in the plant than in the laboratory. Consequently the zinc-iron ratio in the first roughers has not been as high as previously indicated by laboratory tests.

*See January 1952 Bulletin, pp. 16-26; *Transactions,* this vol., pp. 15-25.

DISCUSSION

ON

Mapping Complex Folds in the Slocan Series British Columbia*

By A. B. IRWIN

(*Transactions, Volume LV*, 1952, pp. 242-244)

Discussion at Meeting

Dr. D. Derry (Ventures Limited) referring to the diagram (Figure 15) which shows a transverse structure section across the Central Slocan area, asked what the assumed structure at depth was, remarking that dips are steeper and more uniform at depth and that the contortions with gently-dipping axial planes are at higher elevations. He wondered whether the section represented part of a very large fold with steeply-dipping axial plane.

Mr. J. D. Little (Canadian Exploration, Limited) replied that E. B. Mayo (Kelowna Exploration Company) and others had considered that the entire section illustrated faces to the southwest and represents part of the lower limb of a great overturned syncline which is concave toward the southwest.

Dr. R. DeWit (Geological Survey) remarked that the photograph of a ripple mark (Figure 7) showed internal lamination conformable to the ripple mark and wondered if the structure was not due to slumping during consolidation, since the normal mode of formation of ripple mark would seem to preclude the possibility of such internal stratification.

Mr. Little remarked that in his opinion such structures should not be depended upon alone without other confirmatory evidence of the direction of tops of beds.

Contributed Discussion

Dr. C. E. Cairnes (Geological Survey (1):) I had the opportunity some time ago of looking over a copy of Dr. Irwin's thesis on his investigations in the Slocan, which has formed the basis for the present paper. I was, of course, interested in any contribution on Slocan geology or its mineral deposits, as, many years ago, I had spent several successive field seasons in that widely productive area.

I was particularly interested in Dr. Irwin's discussion of the structural features of the Slocan series of marine sedimentary rocks, in which so many of the mineral deposits occur, and must admit that I viewed with some astonishment his interpretation of the structure of this metalliferous series. Naturally, I had expected that his work on a scale of 1 inch to 200 feet would lead to results more detailed and more precise than the relatively hurried earlier coverage on 1 inch to 4,000 feet, but at the same time I supposed that, in a general way, the structure as interpreted by the late Dr. C. W. Drysdale (1916) and later by me (1925-27) would have an overall resemblance to the more recent efforts by Dr. Irwin.

However, such did not prove to be the case. Structure-sections based on the earlier and the later work suggest a broad similarity between the directions of dip of the strata at the surface. But at depth any such fancied resemblance disappears, and the differences, it seems, may be imputed to the neglect of early workers to recognize and apply the several criteria now in use, and well illustrated in Dr. Irwin's paper, for determining the tops of beds.

Actually, the application of available criteria for the determination of tops of beds has not been a feature of standard geological mapping in the Cordilleran region, though for many years now it has been regarded as necessary practice in unravelling the complicated structures of Precambrian formations in the Canadian Shield. In the mountains of western Canada where, in general, a three-dimensional view has been regarded as an advantage in structural interpretation and where the excessively involved structures of Precambrian terrains are hardly anticipated, it has been, I am sure, more common practice to accept attitudes at their apparent face value — other than, perhaps, to check the direction of tops in vertical or near vertical beds or to suspect the occurrence of overturned strata under special conditions, as in the vicinity of strong reverse faults.

I do not, in any sense, decry the methods adopted in Dr. Irwin's field studies, but I am a little perturbed at the vast difference in results based on earlier and on later, more detailed methods of field research, and am not a little concerned regarding the time that would seem to be required for adequate representation of the structure in these mountain areas. In this particular instance, too, I feel especially concerned at the conclusion drawn by the author that, based on his methods of structural interpretation and "making allowance for the average dip, local thickening, and some repetition of the beds" (an allowance, by the way, amounting to a reduction by about 50 per cent in the apparent thickness), the Slocan group is now to be regarded as a monoclinal, southwest-dipping series with a total estimated thickness, from its base in the northeast, at the Kaslo series contact, to where it is cut off by granitic rocks on the southwest, of some 37,000 feet, as against a previously estimated maximum of less than 7,000 feet.

Many fossil collections obtained from various localities, particularly in the eastern half of its outcrop area, have indicated beyond reasonable doubt that the Slocan series is of Triassic age. Furthermore, as we know of no definite

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* See December 1951 Bulletin, pp. 788-795; *Transactions*, Vol. LIV, 1951, pp. 494-5011.

(1) Published by permission of the Director-General of Scientific Services, Department of Mines and Technical Surveys, Ottawa.

Lower or Middle Triassic marine sediments in British Columbia west of the Rocky Mountain Trench, whereas the Upper Triassic is widely represented, it seems highly probable that these fossil collections from the Slocan represent the Upper Triassic epoch.

Now, a brief reference to some of the better known, marine Triassic sections of the world reveal no such thick columns as that postulated by Dr. Irwin for the Slocan. In Spitzbergen, the complete section, comprising Lower, Middle, and Upper Triassic series, is only about 3,600 feet thick. In the Bavarian Alps and Tirol and in the Lombardic Alps, where complete sections have been carefully measured, total thicknesses reach 7,650 and 8,850 feet respectively, of which about half is assigned to the Upper Triassic. In the western United States (Oregon, Idaho, California, and Nevada) the Triassic system is represented by marine calcareous deposits aggregating approximately 4,000 feet in thickness, of which about 2,000 feet is assigned to the Upper Triassic. Finally, in northeastern British Columbia, where again the entire system is well represented, measured thicknesses to date indicate a variation from place to place from a minimum of about 4,500 feet to a maximum of about 8,000 feet, of which a maximum of about 3,000 feet might be assigned to the Upper Triassic.

With the above figures in mind, it has seemed to me that my early estimate, based on my structure-sections, of 6,800 feet as a maximum thickness of the Slocan series was probably overly liberal, especially as the series includes as much as 2,000 feet or more of limestone, and otherwise consists dominantly of argillaceous to finely quartzitic beds — sediments that are not characteristic of rapid processes of accumulation. On this basis, and with the probable age of the series in mind, the established thickness of 37,000 feet, or about 7 miles, proposed by Dr. Irwin would seem almost incredible, especially as this is, apparently, not the entire thickness, for the upper, southwest contact of the series is against intrusive granitic rocks of the Nelson batholith.

Dr. Irwin, of course, was not particularly concerned with the age of the Slocan series in his structural interpretation, and in fact found nothing either to support or reject the earlier determination. Nevertheless, the problem is still there. If the Slocan series has the vast thickness he ascribes to it, surely it must represent much more than Triassic, and especially Upper Triassic. time. Does this mean that our fossil evidence is incomplete or unreliable, and that perhaps great thicknesses of late Palaeozoic and (or) post-Triassic Mesozoic rocks are represented in this thick sedimentary column? Or could it mean that there are major repetitions in this series that were not recognized, and that, in consequence, its thickness is much less than estimated? On the basis of available palaeontological evidence, regional structural relations, the records elsewhere of marine Triassic sedimentation, and the lack of any observable unconformities within the Slocan series, I should be greatly inclined to suspect the latter alternatives.

Author's Reply

In reply to Dr. Derry: The structure at depth is open to considerable speculation. The bedding attitudes in this area are generally moderate to steep, whether overturned or not, except for the Idaho Peak recumbent folds, so structures have been projected to depth in Figure 15 in accordance with these observations. Mapping by Dr. H. M. A. Rice east of Kootenay lake indicates that the Slocan section is part of the west limb of a major anticline extending from Slocan lake 70 miles eastward to the Rocky Mountain Trench, with a core of early Proterozoic rocks centred in the Purcell mountains west of Kimberley.

In reply to Dr. De Wit: The ripple-mark shown in the photo (Figure 7) is a part of a remarkably uniform structure in even-bedded detrital limestone dipping 30° eastward. A number of adjacent beds also display well-developed ripple structure, all of which indicate the beds to be overturned. Cross-lamination in contiguous beds confirms this conclusion. The ripple mark taken as a whole is quite symmetrical and has a consistent wave length, so there is little possibility of slumping during consolidation being an important cause of such a structure. The upper definable limit of the ripple-mark, stratigraphically speaking, is overlain by detrital limestone in which there are irregular and discontinuous wisps of dark argillaceous limestone, similar to the partings between laminae in the ripple-mark.

It appears that there has been a rhythmic deposition of white granular limestone and argillaceous limestone, which was subjected to extreme agitation by oscillation waves in shallow water prior to consolidation. The upper laminae were disrupted while ripples were impressed on deeper laminae by differential compaction and/or intergranular movement towards crests.

In reply to Dr. Cairnes: The comparisons made by Dr. Cairnes of Upper Triassic rocks in different parts of North America and Europe is indeed a valuable contribution to the understanding of the stratigraphy of the Slocan series. I am not qualified to comment further on this aspect. However, as my estimate of a 37,000-foot thickness is subject to considerable scrutiny, I wish to take this opportunity to present additional detailed information upon which the estimate is based.

By traversing along Silver ridge from near the head of Sandon creek westward to Wild Goose peak, I estimated that, in this section alone, 9,000 feet of Slocan sedimentary rocks were exposed, comprised of: 1,800 feet of argillite; 4,100 feet of mixed types including argillite, quartzite, and limestone; 1,600 feet of quartzite; and 1,500 feet of limestone. This section, which was well-exposed, showed by top determinations that 95 per cent of the beds face west. Allowances were made for dip, obliquity of section, and minor repetition in computing the 9,000-foot thickness.

This well-exposed ridge section constitutes only a fraction of the full width of the belt occupied by the Slocan series, but unfortunately other sections compare unfavourably in the amount of good exposure. West of Wild Goose peak, criteria for top determinations were only rarely observed, but all indicated beds face west. Concerning Mt. Payne, lying east of the section. Dr. E. B. Mayo reports (personal communication) that the main structure is a west-facing open anticline with a nearly horizontal axial plane. Farther east again, in covering a section from Zincton to Rossiter creek in the basal part of the Slocan series, Dr. M. S. Hedley reports (B.C. Department of Mines Bulletin No. 22, p. 14):

"Although no obvious major folding into anticlines and synclines exists, a careful search was made for evidence of isoclinal folding in the two sections studied, namely, on the road and railway and on the Jackson road, but none was found . . . It is concluded that no major isoclinal folding and no thickening

of the section by wholesale duplication have taken place, but there is abundant evidence that the geological section has been greatly thickened by minor structures. The amount of the thickening can not be measured".

As the regional dip here is southwestward, and no mention is made of overturning, it may be inferred that beds face west in conformity with sections in the upper part of the Slocan series.

In view of the above information and observations made in the Howson Creek basin and elsewhere, a preliminary estimate of 37,000 feet for the Slocan series seems justified. However, there is no doubt in my mind that further detailed structural and stratigraphic studies will result in more reliable estimates which may differ greatly from that above.

The Rod Mill in the Sullivan Flow

By H. R. BANKS *

(*Ottawa General Meeting, January, 1952*)

(*Transactions, Volume LV*, 1952, pp. 245-251)

ABSTRACT

The rod mill introduced into the Sullivan Concentrator flow in September, 1948, replaced two sets of 74 in. by 20 in. rolls and their accompanying screens. In addition to the work formerly done by the rolls, the rod mill is producing a finer product and one which provides more suitable feed for the primary ball mills. The object of the finer discharge from the primary mills is to make possible a feed directly from the mills to a series of flotation machines where a large portion of the galena can be withdrawn from the circuit while the galena is still coarsely granular. The interposing of classifiers between grinding and flotation means the selective grinding of the galena.

The author summarizes the work of the rod mill through the first 5,000,000 tons and shows that the wear of lifters, liners, and rods is definitely below normal for the type and size of rock being handled.

The rod mill, driven through a fluid coupling, has been subject to a range of speeds from 13.5 to 19. A pinion in the speed reducer was changed to allow the extending of the normal speed range of the fluid drive of approximately 4 r.p.m.

Since the critical speed inside the liners is 23.4, the range in per cent of critical is 57.7 to 81.2 per cent.

The capacity of the mill increases with increasing speed but the character of the product remains approximately the same except when the feed rate is too high.

Consumption of steel, power, and other operating data are presented.

The author introduces certain fundamental observations on the principles involved in comminution in order to support the special design and operation of the rod mill.

* * *

THE INTRODUCTION of the large rod mill into the flow at the Sullivan Concentrator of the Consolidated Mining and Smelting Company of Canada, Limited, Chapman Camp, B.C., took place in September, 1948. This equipment replaced the two sets of 20 in. by 74 in. rolls with accompanying screens and conveyors.

*Superintendent, Sullivan Concentrator, The Consolidated Mining and Smelting Company of Canada, Limited.

Before proceeding with the presentation of the operating data, I would like to offer certain personal reflections on the general theme of comminution. These were drawn upon in the original concept of a machine to handle the feed to that portion of the Sullivan Concentrator which follows the sink-float operation.

Comminution begins with the start of the first drill hole in the face of the drift or stope where the ore is mined.

The mechanical energy applied through the rock drill and later through the explosive force of the dynamite results in a breaking down of the ore in place into pieces sufficiently small to be handled by the first in a series of machines used to prepare the ore for the process of concentration.

That size control can be effected in blasting has always been known. To the mucker forced to sledge a large portion of the ore through a grizzly, size control in blasting is the *most* important part of the operation.

I do not propose to introduce any highly technical data which would atempt a control of the application of the power equivalent of the explosive force toward comminution. It is interesting to note, however, that based on the heat equivalent of the burning of one pound of nitroglycerine, a force of 0.98 h.p. hour is generated. Since the actual explosion time is of the nature of 1/8000th of a second, the total available energy released in that extremely short period of time is the equivalent of approximately 36 million horsepower!

Since comminution is fundamentally the application of potential energy toward the fragmentation of particles in a series of decreasing size ranges, I feel that one is justified in examining this blasting step in comminution in terms of relative efficiencies because of the enormous energy involved.

If the breaking down of ore in the mine can be controlled as to size to avoid distasteful physical energy, is there not a distinct possibility that a further study of drilling and blasting as a comminution step might prove of value? One can hardly look forward to the day when the miner will present to the millman the ore completely prepared for flotation or other method of concentration. There is some possibility, however, that the recognition of differences in fragmentation brought about by new techniques in drilling and blasting may lead at least to a considerable reduction in the duty of the initial crushing equipment.

The actual amount of work input needed to break a two-foot cube into sixty-four six-inch cubes is not great as measured by the new area developed. Yet, to apply that energy, massive machinery must be set up and the handling of coarse lumps of ore inevitably results in heavy wear.

The crushing tests by pressure and impact carried out by the Allis Chalmers Company under the direction of Fred C. Bond and published in 1945 offer much food for thought. These are described in *Technical Publication* No. 1895 of the American Institute of Mining and Metallurgy.

This type of basic study should eventually lead us into a more efficient application of the available energy in our problem of comminution.

The most striking thing about the series of tests referred to is that the force was gradually increased until rupture was effected. As in the case of the sledging of a rock, there is a zone where the work effort is entirely absorbed by the elasticity of the rock specimen. But on just emerging from that zone, even by an infinitely small increment, the work is done. The potential energy built up in the contracting of the rock within its elastic limit is brought to the point where it is greater than that elastic force and the release of the pent-up energy effects fragmentation.

To illustrate the point further, one can visualize the turning of a ball-mill filled with rubber balls. The useful work in such case would be zero, since the elasticity of the balls would completely resist any tendency for fragmentation.

It is axiomatic that in all types of crushing and grinding equipment it is necessary to apply a force which will be sufficient to overcome the elasticity of the rock particles.

A source of inefficient application of mechanical energy is where the machine is actually too heavy for the work to be done.

The entire field of comminution has always been a challenge. In each step there are basic forces to be overcome.

In Figure 1, I have attempted to illustrate the point of more efficient usage of mechanical energy. The stamp on the left is a typical stamp with a layer of feed on the die and the head about ready for the drop. If the release of potential energy of the stamp is not great enough, the crushing step will fail to accomplish any useful work. If, however, the design of the stamp is sufficiently heavy and the drop is great enough, the release of force will be sufficient to effect useful work. To attain this latter result there must be a definite margin of energy above that actually required. The excess is absorbed in the die and its supporting foundations.

At the right is shown a hypothetical series of dies. That energy which is left over when the shoe strikes the first layer of ore in the upper die would pass through to the second layer and so on, thus expending the excess in useful work rather than having it absorbed by the foundations.

This is actually the effect for which we are striving in the rod mill. In fact I have felt for some time that the high-speed rod mill is the modern counterpart of stamps.

Machines for comminution are of three types:

(*A*) Those in which the mechanical energy is applied to wearing faces which are moved mechanically and which squeeze or nip the feed and break it down. Examples are jaw crushers, gyratories, rolls, etc.

(*B*) Those machines in which the grinding or work media are charged with potential energy which is released under the force of gravity. This group comprises stamps, ball mills, rod mills, etc.

(*C*) Those machines in which the potential energy is imparted to the feed itself and is released by impact as in cylindrical mills with autogenous media or by devices using the principle of impingement against a wearing plate.

The machines in type *A* are usually ponderous and the power to drive them while empty is a fairly high percentage of the power used while crushing ore. The ratio of potential energy to that theoretically required is high so that it is obvious that the mechanical efficiency is low.

In type *B*, the means employed to invest the work media with potential energy are varied but the fundamental principle of lifting against the force of gravity and then allowing gravity to drag the work media down with the feed interspersed is common to all. The cylindrical drum provides a very convenient scheme for imparting potential energy to the work media. A rising conveyor would also serve to pull the work media against the force of gravity. If the conveyor were provided with prominent cleats, and if immediately above the loaded conveyor was a mass of work media falling onto the conveyor then travelling down it in the opposite direction, a condition similar to that existing in the ball mill or rod mill would result. It is obvious, also, that as the conveyor was speeded-up the sharpness of the blows exchanged between the up-going and the down-coming work media would be increased.

That, to me, is the main feature of the high-speed rod mill. The prominent lifters represent cleats on the belt sufficiently high to prevent slippage, and the speeding-up of the elevating mechanism introduces certain phases of the type *A* equipment, since the up-going rods backed by the lifters provide a striking mechanism and definite impact.

Type *C* offers such a vista of conjecture that I hesitated to refer to it. To complete the classification, however, it was necessary. It is in that field where I look for the solution of the problem of ultra-fine grinding.

To return to the Sullivan rod mill: we felt that, by driving the upprising rods at a comparatively high speed, we would set up a stamp action and effect the preliminary coarse breaking. We realized that if the mill speed was too great there was a danger of introducing slippage between rods and liners. This made the prominent lifters necessary. The fact that some rods leave the mass at the apex of the mill and fall freely against the side of the mill did not present any great difficulty. At the point of impact, the mill and the rod are travelling approximately in the same direction and at similar speeds.

To illustrate the rod action, we took pictures of a scale-model rod mill with lucite ends running at the same degree of critical speed as is the Sullivan rod mill. (For the picture shown in Figure 2, the camera speed was cut to 1/50 second and the rod ends (actually welding rods)

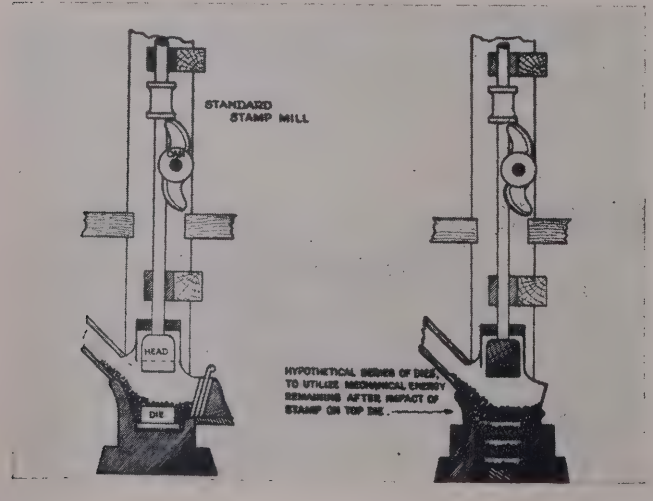
STANDARD STAMP MILL
HEAD
DIE
HYPOTHETICAL SERIES OF DIES TO UTILIZE MECHANICAL ENERGY REMAINING AFTER IMPACT OF STAMP ON TOP DIE

Figure 1.

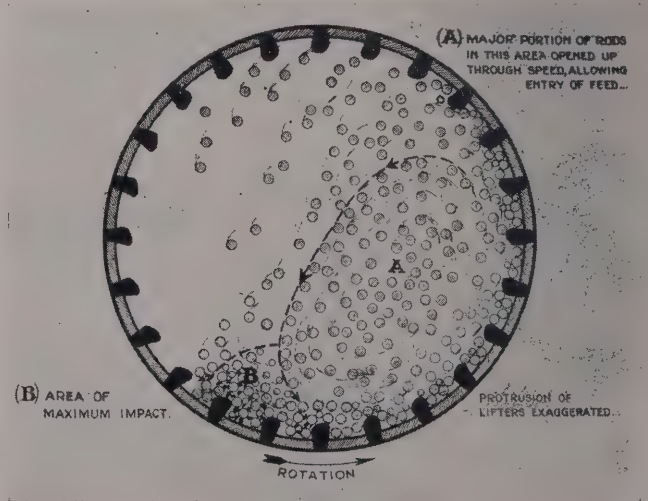
(A) MAJOR PORTION OF RODS IN THIS AREA OPENED UP THROUGH SPEED, ALLOWING ENTRY OF FEED...
A
B
(B) AREA OF MAXIMUM IMPACT.
PROTRUSION OF LIFTERS EXAGGERATED.
ROTATION

Figure 4.

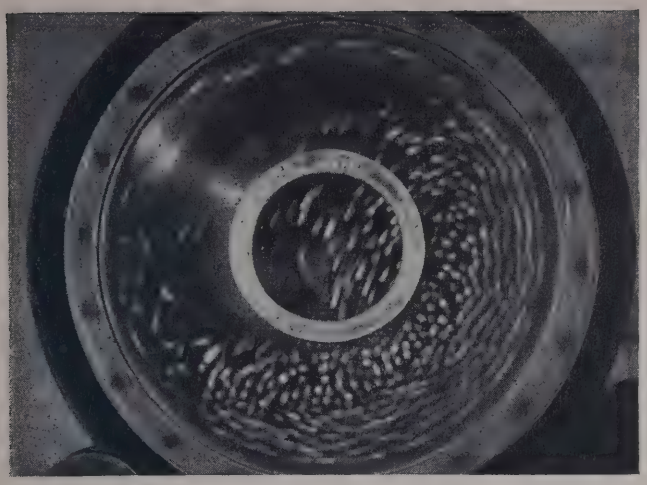

Figure 2.

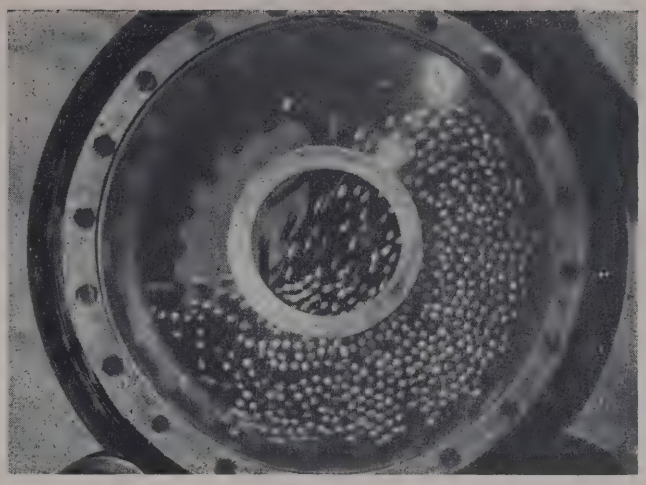

Figure 3.

were buffed to reflect the light. Granted there is no feed in the mill, yet the travel of the rods is very evident.

To clear up the point concerning the impact of the falling rods on the mill shell, you will note that the trace of the falling rods is about equivalent to that of the rods against the rising shell. The Sullivan rod mill has a particularly noticeable lack of 'bump'. Those rods which strike above the pulp line do so with a fairly gentle blow. One recalls the old-time movie when the rider leaves his horse and jumps to the train. It was essential that horse and train be moving in the same direction and at approximately equal speeds!

The camera speed for Figure 3 was 1/400th second. The noticeable feature is the looseness of the rod charge. This is essential if the feed is to enter the charge to obtain maximum mill capacity. When one realizes that feed, up to 2-inch in size, is passing into the mill at the rate of 5½ tons per minute it is very evident that the passageways must be generous in cross-sectional area.

If a tonnage greater than the capacity of the mill at any given running speed is fed, the rod charge will move forward. The rod load will bunch up with unground feed and rods will actually be forced out of the mill. The remedy is to cut off the feed entirely and resume feeding at a reduced rate or speed up the mill.

Figure 4 illustrates the principles involved in the rod mill. The protuberance of the lifters is purposely exaggerated. The area of maximum work is indicated. It is best, I believe, to refer to this area as that of maximum impact because work in the finer ranges mounts so rapidly when measured in terms of new area produced.

Prior to the major flow changes introduced during the past few years at the Sullivan operations, the crushing equipment consisted of one jaw crusher underground for the lower level ore, followed on the surface by two jaw crushers in parallel, three gyratories, also in parallel, one seven-foot standard Symons crusher, and two sets of 74 in. by 20 in. rolls in closed circuit with six 4 ft. by 5 ft. Hummer screens.

The present plant is underground and consists of one jaw crusher (36 in. by 48 in.) and two 7-foot standard Symons crushers. Scalping screens are operated ahead of the Symons crushers.

The product of the underground crushing plant is the feed to the sink-float operation at the Concentrator.

The rolls are no longer a part of the regular flow but can be used in an emergency.

To locate the function of the rod mill in relation to the original flow, a comparative screen analysis of feed and discharge of rolls and rod mill is presented in Table I.

This tabulation illustrates that the rod mill is actually doing some of the work of the original Symons crusher, all of the work of the rolls and screens, and a portion of the work done previously by the primary mills.

One of the objects of obtaining a finer discharge from the primary mills was to permit the application of flotation to the primary mill discharge. The purpose of this move is to recover a portion of the galena in a more granular state than where grinding is completed before submitting the pulp to flotation.

A comparison of the screen analysis of the primary ball mill discharge before and after the introduction of the rod mill is shown in Table II.

The rod mill was supplied by the Hardinge Company and was built in the John Inglis shops in Toronto. It was designed to handle 400 tons per hour.

Figure 5 shows the mill shell as it was received. Dimensions and other data are as follows:

TABLE I.— Screen Analysis of Feed and Discharge of Rolls and Rod Mill

	+1″	−1″ +½″	−½″ +4	−4 +10	−10 +48	−48 +200	−200
Feed to rolls	1.4	33.8	35.2	14.1	6.9	3.6	5.0
Feed to rod mill	21.3	14.5	34.0	7.4	8.2	10.6	4.0
Discharge of rolls			5.8	44.9	24.3	10.1	14.4
Discharge of rod mill			4.8	28.4	36.7	14.8	15.3

TABLE II.— Screen Analysis of Primary Ball Mill Discharge Before and After Introduction of Rod Mill

	+6	+10	+48	+100	+200	−200
Before	0.4	1.6	21.7	39.2	53.7	46.3
After	—	0.4	12.4	32.4	51.4	48.6

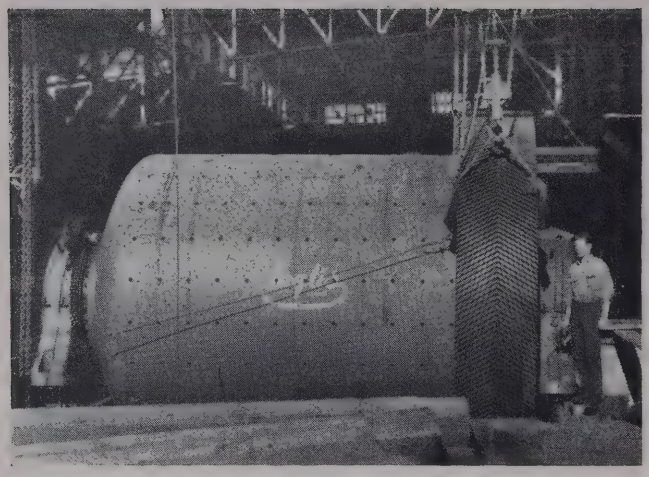

Figure 5.

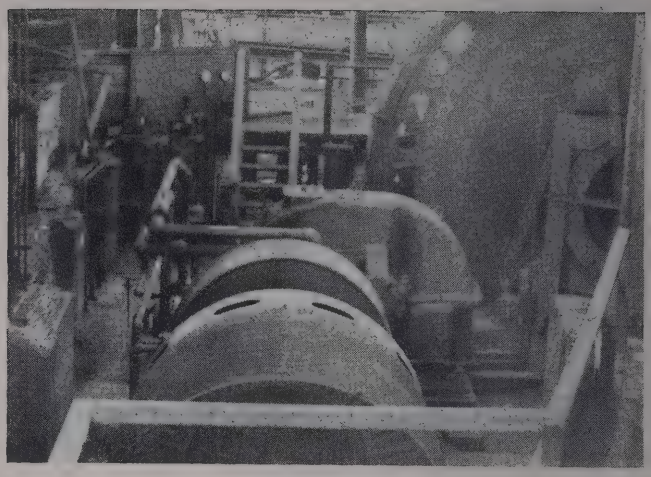

Figure 6.

Outside shell diam.	11′5″
Thickness of shell	1¼″
Inside brl. liners (new)	10′8″
Inside wedge lifters	10′1″
Trunnions inside diam.	3′8½″
Feed trunnion opening	25″
Disch. trunnion opening	40″
Length of mill (cylinder)	12/0″
Length of mill (trunnions)	18′6″
Volume of mill	1,040 cu. ft.
Weight of liners	46 tons
Wt. shell and trunnions	30 tons
Weight of ring gear	10 tons
Weight of rod charge	76 tons
Total revolving weight	162 tons

The general arrangement of the rod mill is shown in Figure 6. The mill is driven by a 1,000 h.p. synchronous motor made by the Westinghouse Company. The speed of the motor is 720 r.p.m.

A fluid drive coupling made by the American Blower Corporation permits a speed variation down to 78 per cent of the top speed.

The coupling is size 48, type *P.E.* The need for this coupling is that the starting load of the rod mill is too great for the power supply if directly off the line.

The ability to change the speed has been useful in developing the capacity of the mill. Through the use of the coupling and two alternative reduction ratios available in the Dominion Engineering Company speed reducer, it has been found possible to operate the mill through a range of speeds from 13.5 r.p.m. to 19 r.p.m. This range is equivalent to a range of from 57.7 per cent to 81.2 per cent of critical.

The advantages possible through a fluid drive coupling to vary the speed of a rod mill to meet changes in feed characteristics or tonnage requirements would have to be weighed against the power costs involved. The variation is possible through slippage which, of course, is wasted power.

It has been our experience that varying the speed permits a variation in capacity but does not change the product materially. Therefore, where the tonnage and ore character are constant, the mill should be run at a set speed. The highest efficiency in power usage with the fluid coupling is at the top speed. To operate the fluid coupling continuously at less than top speed in order to be in the position to increase the mill speed to meet emergencies would demand a study in economy.

The Dominion Engineering Company speed reducer originally supplied had a ratio of 3.444 : 1. Later, when it was found that the mill capacity could be increased with increasing speed, the reducer ratio was changed to 3.027 : 1 through the purchase of a new pinion.

The ring gear of the mill has 278 teeth and the pinion 24 teeth. The gear faces are 32-inch and are cut herringbone.

The trunnion bearings are two micarta half-bushings for 48-inch diameter, 15-inch wide, main bearings. The material from which these are made is laminated linen impregnated with plastic and compressed.

The lubricant is water. Some oil is used when starting the mill and a few drops of oil are added just as the mill is stopped.

The water for lubrication is filtered, though our experience indicates that the bearings will accept small amounts of silt. They performed satisfactorily through a period when the water was none too clear.

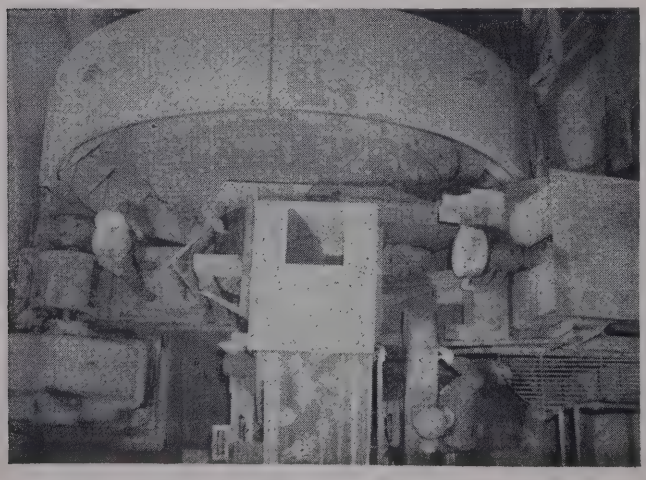

Figure 7.

Figure 8.

Figure 9.

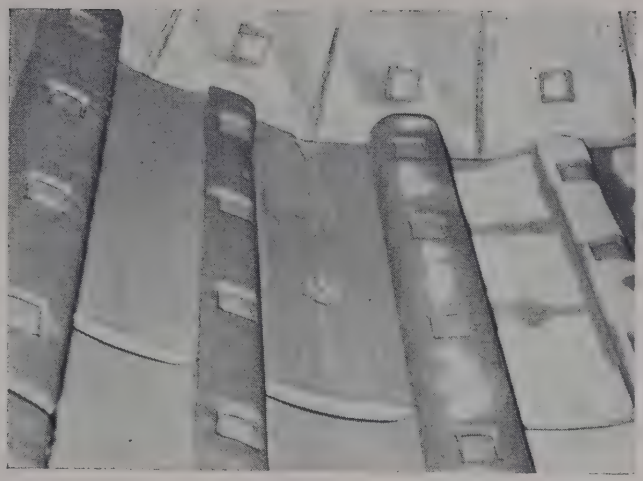

Figure 10.

Feed to the rod mill is from the sink-float plant. The undersize of the preparation screens (¼-inch mesh) is pumped to a launder delivering the pulp to a dewatering screw classifier. The sink is drawn from a storage bin.

Figure 7, a view looking down at the feed arrangement, shows the two sources of feed. The tonnage of undersize from the preparation screens is approximately 3,000 tons per day. The balance is drawn from the sink storage bin by feeders and conveyors. The two products are joined at the rod mill feed chute and total approximately 8,000 tons per day.

The rather large feed trunnion opening permits the use of a simple chute, a decided blessing.

The coned feed end and the rod charge well opened up through the high speed makes the feeding of the mill a simple matter (see Figure 8). A spill drum protects against spillage from the occasional surge.

Figure 9 illustrates the lining of the rod mill when new. The balls, placed to show the size and spacing of the lifters, are 3 inches in diameter. The rotation of the mill is clockwise from this view.

The lifter bars are 6½ inches thick and project 3½ inches above the face of the barrel liners. They are bolted to the shell. The width of each barrel liner is 13 inches and the length 3 feet.

Since it was assumed that the lifters would bear the brunt of the striking blow, they were purposely designed with a protruding leading edge. Manganese steel was chosen as the metal to be used because of its work-hardening property.

For the barrel liners, *Ni-Hard* was chosen, assuming a need for wear resistance. The feed end liners are of manganese steel, guarding against a direct blow when charging the rods, while the discharge end liners are *Ni-Hard*.

The feed and discharge liners have stood up remarkably well considering they have been subjected to glancing blows from the rods at times when a tendency for the rods to shift lengthwise was experienced.

Single-ply soft rubber launder lining, ¼-inch thick, was cemented to the shell, laid with the fabric next to the liners.

There has been one change of lifter bars since the mill went into operation. During the life of the first set, the mill handled 3,973,106 tons. The average life in operating days was 536 and the consumption was 0.00793 lb. of lifter bar per ton of feed.

The original barrel liners are still in service, as are the feed and discharge liners. A conservative estimate of the life of these is 6,500,000 tons of feed for the end liners and 7,300,000 tons for the barrel liners. On the basis of these estimates, the total consumption of liners and lifters should approximate 0.0168 lb. per ton.

Figure 10 shows the mill with the first change of lifters half completed and the original wedge bars. The wedge bar is identical to the lifter bar but is used to hold the barrel liners to the shell and does not sit on the shell as do the alternate lifter bars. Since the wedge bars projected slightly farther out than the alternate lifters, the latter wore down first and were changed. There was a definite speeding-up in rate of wear during the period that the lifters had reached the stage for renewal. It is, therefore, evident that slippage increases below a certain lifter height.

For the first set of lifters, the following edge stood too straight and a pocket resulted between lifters. The protruding leading edge increased this tendency. As a result, several of the spaces would become wedged with rods. When the rods remained in this position for some time, evidences of wear through slippage would show up; the rod would show a flat side.

The replacement lifters were designed with a smaller radius in the following edge and the condition has been largely corrected.

A normal rod load is 76 tons. This has been varied from 65 to 96 tons, or, in terms of mill volume, from 36 per cent to 53 per cent.

At a speed of 19 r.p.m. and with a load of 76 tons of rods, the horsepower needed to drive the mill is 1,000. Additional rods at this speed would demand more power, a risk we do not wish to take.

The size of rods has been the subject of much investigation. It was natural to assume that a heavier rod would strike a harder blow. Therefore, provision was made for rods up to 4½-inch diameter. This was wrong. The larger rods did not give as good a product as the smaller rods. The discharge was coarser and the rejected broken rods were of greater diameter.

Apparently, 3½-inch rods are the most suitable for the present type and tonnage of feed.

Figure 11 shows the rod load immediately after a new charge has been placed. The peening of the rod ends as they strike the discharge liners is noticeable. It is also to be seen that there is very little tendency for the ends of the rods to wear to a point nor for the rods to flatten.

The rods are manufactured by the Steel Company of Canada and are of the customary analysis:

Carbon	0.9 per cent
Sulphur	0.035 "
Silicon	0.25 "
Manganese	0.75 "

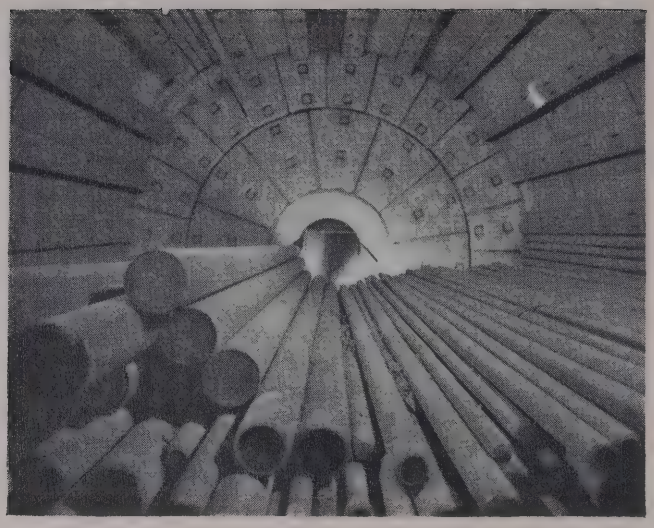

Figure 11.

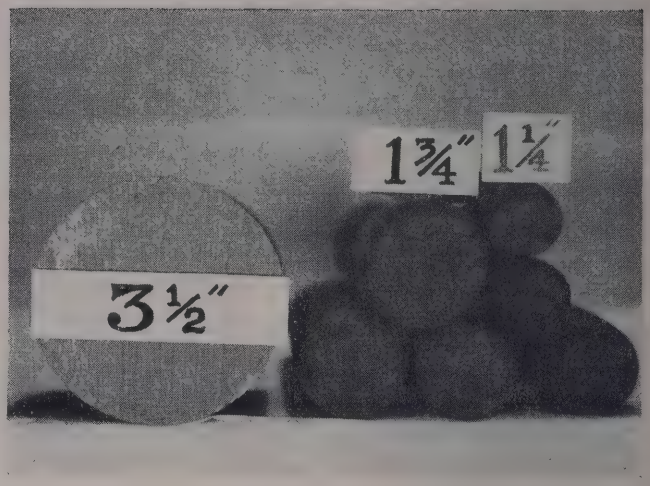

Figure 12.

Typical rejected rods are shown in Figure 12. The rods are rejected in varying lengths which break off and at sizes which appear rather disturbing until one realizes that the consumption, based on 4,600,000 tons of feed, has been only 0.354 lb. per ton.

A side view of a rejected rod (Figure 13) shows a slight tendency toward wearing to a tapered end. This is rather unusual.

The rods break off and the pieces work out to the dished discharge, eventually tumbling on to the trunnion liner and thence to the trash grizzly, from where they are picked off.

The new rods were charged weekly during the previous working schedule of five days per week. With the present continuous operation, the mill is stopped long enough each day to charge the rods. This should actually result in better grinding since the average rod load is higher than under the weekly charging routine.

It is evident from the screen analysis of the feed and discharge that the principal work zone of this rod mill lies in the plus-10-mesh size.

The effect of mill speed on capacity has been rather difficult to tie to a specific curve because our testing has been done only when mill feed conditions would permit. The first obligation has been to keep abreast of concentrator demands.

Table III is based on the data collected during the operation of the rod mill. It must be understood that variations in feed characteristics must be taken into account but, having made due allowance for these, the varying speeds and equivalent tonnage are as given in the Table.

When it is realized that the discharge does not vary appreciably in screen analysis, the increment in work done with rising mill speeds is startling.

Pulp density was, naturally, the subject of much testing. The general conclusion was that the optimum density lies between 80 and 85 per cent.

Actually, more grinding work can be done with a pulp above 85 per cent solids but, if the feed is coarse or if tonnage is being stressed, the higher density pulp will contribute to oversize.

That, briefly, is the story of the rod mill in the Sullivan flow.

I am grateful to Cominco Management for the opportunity of presenting to the C.I.M. a description of the mill and its performance since its installation and through the first

TABLE III. — Rod Mill Operation Data

Speed (r.p.m.).....	13.5	14	15	16	17	18	19
Tons per 24 hr.....	5,600	5,850	6,580	7,170	8,255	8,820	9,375

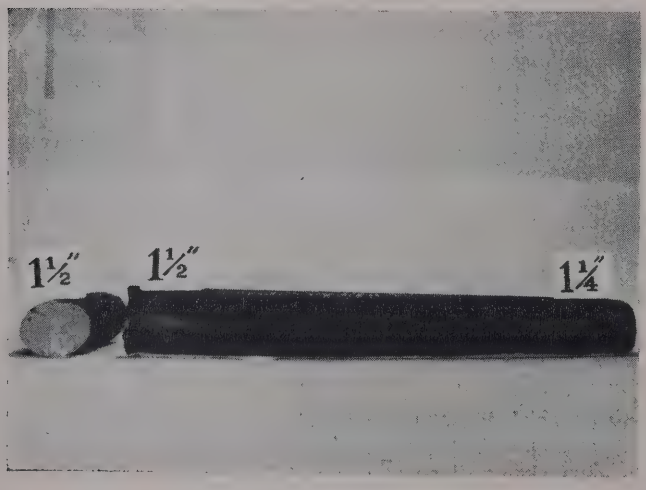

Figure 13.

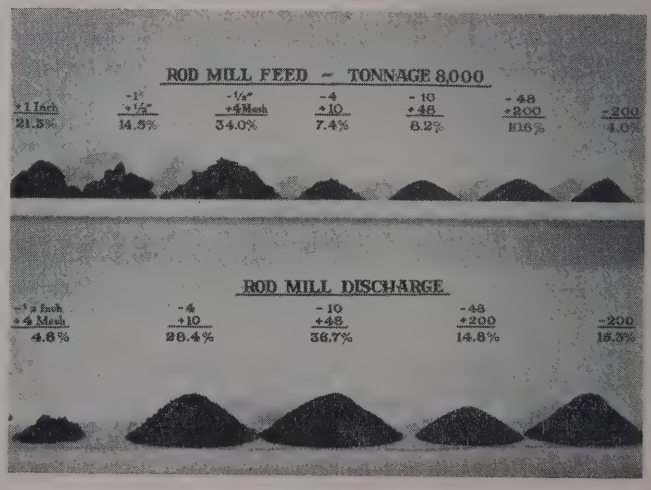

Figure 14.

five million tons of feed. I wish to acknowledge the excellent work of the late G. J. Knighton under whose direct guidance the rod mill went through its growing pains.

Great credit is also due to the staff of the Sullivan Concentrator, under whose constant attention the rod mill continues to prove its value.

I wish also to acknowledge the fact that it was through J. F. Myers, of the Tennessee Copper Company, and E. H. Rose, formerly of the International Nickel Company, that we were encouraged to try the high-speed rod mill.

There is still another acknowledgment that I would like to make at this time and that is to the C.I.M. We, who have devoted our lives to the economic winning of our national mineral wealth, are entitled to recognition as members of a profession. It is through the auspices of the Canadian Institute of Mining and Metallurgy that such recognition can come. It is through the Branches of the Institute, spread across the country, that contact with the profession can be maintained. It is through the discussions, first within the Branches and later through the National Headquarters, that technical progress in our profession can be achieved.

I sincerely believe that the C.I.M. can and will, with the help of the individual members, serve this profession of ours in an ever expanding sphere.

* * *

Contributed Discussion

Mr. B. S. Crocker (Lake Shore Mines, Limited): As this mill is one of the largest rod mills in the world, we have all been waiting with considerable interest for the first published information regarding its operation. I was particularly interested, as I was present at Kimberley in the fall of 1946 when Bert Banks held an informal discussion with several prominent rod mill operators regarding the final details of this installation. A mill of this size presented several problems which were discussed at that time.

One of the main features of this report is the relatively high percent critical speed at which this mill is operated. They are doing very coarse grinding with a feed 2 per cent plus-1-inch and a discharge 5 per cent plus-4-mesh. For this coarse grinding they ran the mill from a 58 per cent critical speed up to 81 per cent critical speed. They obtained their best operating results at the highest speed. When their tonnages are plotted against the speed of the mill, the resulting line shows about 10 per cent more capacity at the highest speed than would be expected if the capacity were assumed to vary directly as the speed. However, as full screen analyses, power and road load are not given, it is not possible conclusively to prove this point. The fluid drive, no doubt, would spoil the chances of getting a reliable power figure at the slower speeds.

At Lake Shore we, also, are using a high-speed short-length rod mill doing a similar job to this mill at Consolidated. Our mill is 7 ft. dia. by 6 ft., which makes it about one-seventh the size of the Consolidated mill. We handle feed which is 7 per cent plus-½-inch (on a very hard siliceous ore on which a 4-inch ball will not handle plus-⅜-inch material) and grind 750 tons per day to 7 per cent plus-6-mesh. We started the mill with about 65 per cent critical speed but could not obtain satisfactory results until we speeded-up the mill to 76 per cent critical speed. Now we have one mill at 76 and one at 81 per cent critical speed. It is remarkable the amount of tonnage that can be put through these short rod mills and still do a good job on the top meshes.

We have been doing some work lately on the power consumed by rod mills and ball mills. When load, percent critical speed, and liner thickness and shape are corrected for, we found that the Consolidated mill should draw 1,010 H.O., an error of 1 per cent from the actual. This is further proof, on a large-diameter mill, that the power varies as the diameter is raised to the 2.6 power.

Mr. Banks reports testing 4½-inch rods and 3½-inch rods, and that he found 3½-inch to be the better, particularly as regards the rejected broken rods. We tested 2½-inch rods against 3½-inch and found little difference in the discharge analyses or in capacity. At light tonnages, the 2½-inch rods were 10 per cent the better, but were worse at high tonnages. The deciding point was that the 2½-inch rods would not break themselves up, and discharged from the mill in pieces two feet long. The 3½-inch rods would not discharge pieces longer than four inches. We have finally settled for 3-inch rods, with satisfactory results.

Some one asked regarding the length of rods in a mill. Mr. Banks reports his are 6 inches shorter than the shell length. To get maximum capacity in a 6-foot mill we use rods only 2 inches shorter than the shell length. We have also successfully used *Ni-Hard* liners without backing whatsoever.

I would like to ask Mr. Banks if they periodically remove small-diameter rods from their mill?

Mr. Banks: No stripping is done.

The Effect of Certain Metallic Impurities on the Properties of Zamak 3-Type Zinc-Base Die Casting Alloys *

By R. C. BELL,† J. O. EDWARDS,§ and J. W. MEIER§

(*Annual General Meeting, Quebec City, Quebec, April,* 1951)

(*Transactions, Volume LV,* 1952, pp. 252-261)

Introduction

THE DIE CASTING industry as we know it today is the direct result of the invention and development of die casting machines during the nineteenth century and of reliable alloys in the twentieth. The early machines were designed to cast low melting point metals such as lead and tin, and their alloys.

It was inevitable that, as these die castings were put into service, the demand should arise for stronger materials. Zinc alloys, often containing lead, tin, copper, and aluminum as major constituents, were relatively easy to cast and stronger than the lead and tin alloys. However, it became apparent after a number of years that these zinc alloys tended to become brittle, distort, and corrode, particularly if kept in warm, humid atmospheres. Thus, considerable quantities of zinc-base alloys were in use before the trouble was realized and effective measures could be taken to determine the cause of the instability.

The deleterious effect of certain metallic impurities on the properties of zinc die casting alloys has now been known for about thirty years. Brauer and Peirce, and, later, other research workers of the New Jersey Zinc Company such as Anderson and Werley, discovered the pronounced effect of lead, cadmium, and tin on the stability of zinc-alloys. At that time, high purity zinc was not commercially available, and work was started to find alloying additions which would prevent the effect of these impurities on the corrosion resistance and dimensional stability of the alloys. It was found that small additions of magnesium improved these properties considerably by hindering the formation of subsurface network corrosion.

It is only since 1930, when the New Jersey Zinc Company commercially introduced their high-purity zinc alloys of the Zamak family, based on 99.99 per cent minimum purity zinc, that zinc die casting alloys have been completely accepted and have won a prominent position in the die casting industry. Strict standardization of these alloys by the American Society for Testing Materials and the Society of Automotive Engineers followed. The question of the actual metal purity required to produce sound die castings is still somewhat controversial, with various opinions being held on the safe limits of harmful impurities allowable. There has been a trend toward a tightening of specification limits for zinc of die casting grade and a demand by many consumers for zinc metal of higher purity than specified by the American Society for Testing Materials.

Many of the opinions regarding metal purity appear to originate from shop practice rather than from any systematic metallurgical investigations. In some instances, the case for very high purity zinc has been based on the frank statement that this would allow for less careful casting techniques, especially in the case of poor die design.

It was, therefore, thought desirable to re-check the effect of the three major impurity elements, lead, cadmium, and tin, especially since the detailed results of previous researches are not generally available.

Objective of Investigation

The primary objective of the programme was to determine as accurately as possible the effect of small amounts of lead and cadmium on the properties of the most widely used copper-free Zamak 3-type zinc die casting alloy. It was desired to ascertain at what level of lead, and under what conditions of ageing, significant detrimental effects could be observed in the standard Zamak 3-type alloy containing varying amounts of cadmium. In addition, tin was included as one of the impurities to be investigated since it has been reported to be very deleterious under some conditions and might make useless a particular alloy containing otherwise harmless amounts of lead and cadmium.

After visiting various die casting plants, it became obvious that a considerable quantity of Zamak 3-type alloy is in use with the 0.1 per cent copper maximum content allowed by ASTM specification. This build-up is mainly due to the re-melting of plated casting scrap, contamination from the Zamak 5-type alloy, and sometimes from the secondary aluminum used to make up the alloy. It was decided, therefore, because of the limited time available, to complete a small programme with 0.1 per cent copper, 0.005 per cent cadmium, and with varying amounts of lead and tin, to determine if the presence of this amount of copper would significantly change the pattern of results established for the copper-free alloys.

Finally, it was decided to check the effect of small quantities of indium on the zinc-base alloys, since it was thought that there might be some possibility of contamination from indium-plated castings or from other sources.

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This investigation was undertaken as a co-operative research project by The Consolidated Mining and Smelting Company of Canada, Limited, and the Canadian Department of Mines and Technical Surveys. The experimental work was carried out in the Physical Metallurgy Research Laboratories of the Department of Mines and Technical Surveys, Ottawa.

Previous Work

Considerable investigation in the field of zinc die casting alloys has been carried out by numerous authors, and a brief review of the more pertinent references is given.

As early as 1917, Williams (1) noted swelling of zinc-base die castings stored in humid conditions. Because the old zinc-base alloys became brittle after ageing at room temperature for some time, considerable research was undertaken in various countries to determine the cause of the embrittlement, and to develop new alloys. Brauer and Peirce (2), and Anderson and Werley (3), were responsible for the work leading to the development of the now standard Zamak type alloys, the patents for which were taken out between 1929 and 1932. During this time, a committee was appointed by the ASTM to investigate zinc- and aluminum-base die casting alloys and this work is reported periodically in the *Proceedings* of the *ASTM* (4). The effect of ageing on the Zamak alloys was reported by Fuller and Wilcox (5).

More recently, the New Jersey Zinc Company published general information summarizing results on the effect of impurities on the modern Zamak-type alloys in their trade brochures 6, 7). Here it is stated that the *ASTM* limits for impurities (Pb 0.007 per cent, Cd 0.005 per cent, Sn 0.005 per cent) represent safe limits for commercial die casting. The maximum limit for lead, based on the amount which will promote the occurrence of subsurface network corrosion, is said to be over 0.007 per cent and under 0.009 per cent. Cadmium is detrimental at some concentrations and neutral at others. Tin, like lead, promotes subsurface network corrosion and the maximum safe limit for tin is given as 0.005 per cent.

In the late 1930's the Apex Smelting Company in Chicago published a number of trade leaflets (*Metalgrams*, Nos. 1-25) giving detailed results on the effect of various impurities and casting conditions on the properties of several zinc-base die casting alloys. The results showed that 0.007 per cent Pb and 0.005 per cent Sn had little effect on the properties of the Zamak 3-type alloy, but did affect adversely the Zamak 5-type alloy. Alloys containing 0.014 per cent Pb and 0.014 per cent Sn were prohibitively bad in both cases.

Detailed Australian work (8) showed that even with 0.009+% lead, properties well inside the *ASTM* specified minima were obtained.

Considerable work has also been carried out, particularly in Germany, on the effects of impurities on the physical metallurgy of the zinc alloys. Much of this work covers alloys other than Zamak 3, but interesting results were obtained on such features as microstructure, mode of corrosion, distribution of impurities, effect of the magnesium addition, etc. (9-16). Additional information on this and other aspects of the zinc die casting field can be obtained from published bibliographies (17).

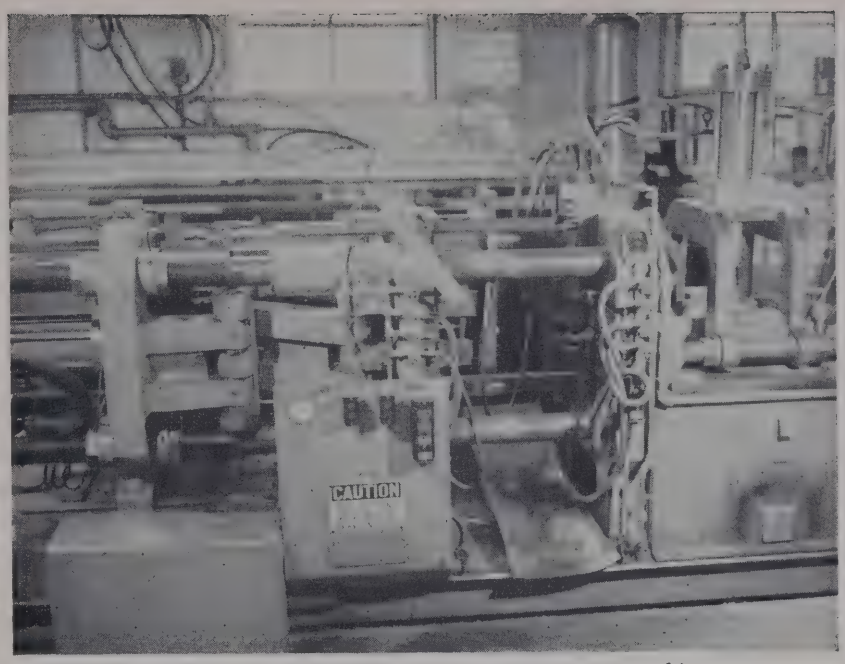


Figure 1.—Side view of Cleveland die casting machine.

The Die Casting Machine and its Operation

The die casting machine used in this investigation is shown in Figure 1. It was manufactured by the Cleveland Automatic Machine Company, and can be used with either a cold or hot chamber by substitution of the necessary parts at the shot end, and some adjustments to the timing and hydraulic mechanisms. Figure 2 shows the shot end of the machine as used for zinc alloys. On this plunger type, hot chamber machine, pressures on the liquid metal of up to 4,800 p.s.i. can be achieved, depending on the diameter of the plunger used. With larger diameter plungers, zinc alloy die castings weighing up to 18 pounds can be produced. Throughout the experimental programme, the machine was operated with a 2¾-inch-diameter plunger, and a pressure on the metal of 1,700 p.s.i.

Figure 3 shows, diagrammatically, the principle of zinc die casting. The molten alloy enters the gooseneck through inlet holes and the hydraulically actuated plunger moves down, forcing the metal into the die. The air trapped in the die escapes through vents in the die until finally these vents are blocked off with metal. The pressure developed by the hydraulic system is then transmitted, through the plunger and liquid metal in the gooseneck, to the metal freezing in the die cavity. Thus, the metal solidifies under pressure. The pressure on the plunger is then released and reversed so that the plunger travels toward the top of the gooseneck, the die is opened, and the casting ejected. The liquid metal in the pot refills the gooseneck as soon as the plunger is clear of the inlet holes, and the cycle is repeated.

The great importance of the die design should be stressed. In all casting methods, gating and venting affects the quality of the product, but in die casting, where metal is cast at high pressures, this influence is especially pronounced. Fig-

*For references, see end of paper.

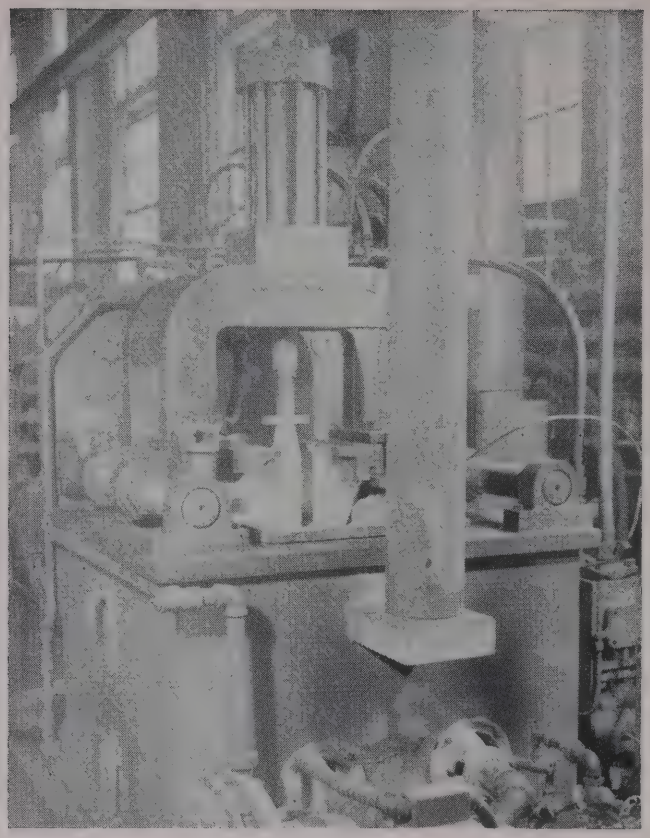


Figure 2.—Shot end view of Cleveland die casting machine.

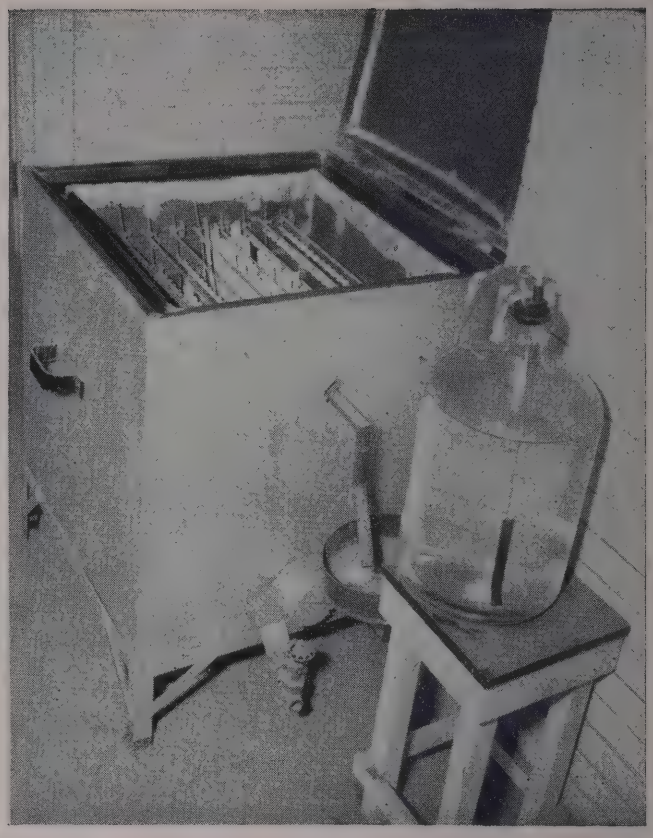
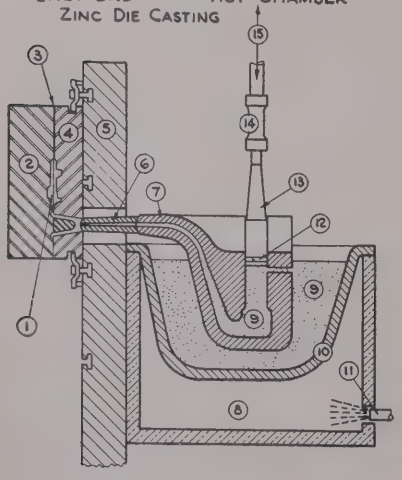


Figure 6.—Steam cabinet.

ure 4 is a drawing of the die used in this investigation. It was based on a design proposed by Werley (18), with some modifications.

Figure 5 shows the ejector view of the test bar die casting, consisting of a sprue, runners, gates, a round tensile test bar on one side and an almost square impact test bar on the other side. In the middle is a waster bar, which is used only to keep the thermal conditions of the die in balance in order to avoid overheating or cooling of die parts. This, in turn, could affect the freezing conditions of the castings. Overflow wells can be seen at the end of each test bar and of the waster bar. Tensile and impact bars conform to *ASTM* specifications, except that the overall length of the impact bar is 9 inches instead of 6 inches. The impact bar thus provides three 3-inch specimens for mechanical testing, which are referred to in this paper as gate end, middle, and vent end sections.

FIGURE 3
SHOT END HOT CHAMBER
ZINC DIE CASTING



1. DIE CAVITY
2. EJECTOR HALF OF DIE.
3. VENTS.
4. COVER HALF OF DIE.
5. STATIONARY PLATEN.
6. NOZZLE.
7. GOOSENECK.
8. MELTING FURNACE.
9. ZINC.
10. CAST IRON POT.
11. BURNER
12. PLUNGER RING.
13. PLUNGER
14. FLEXIBLE LINKAGE.
15. HYDRAULIC SYSTEM.

Figure 3.—Diagram of zinc casting process.

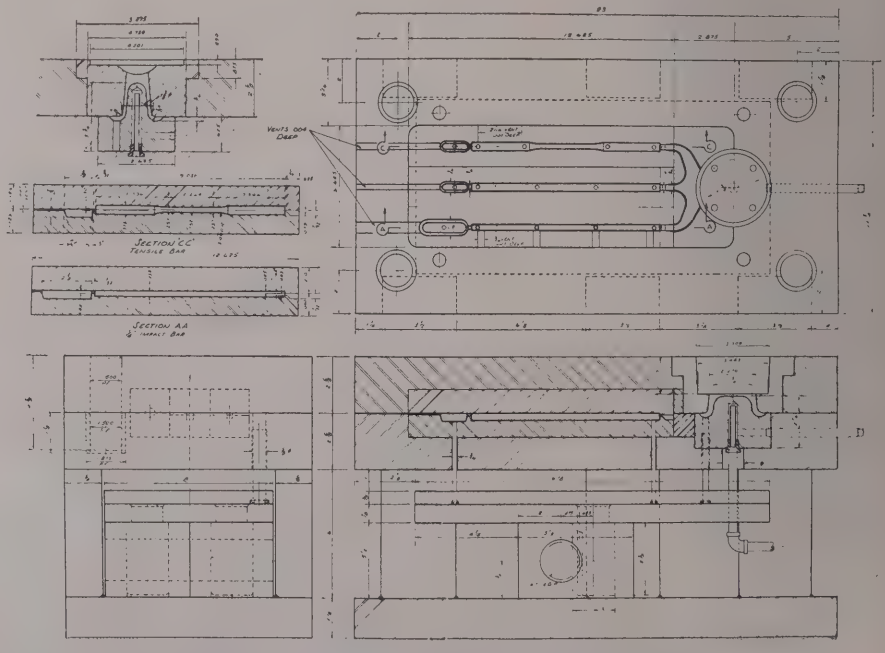


Figure 4.—Drawing of three-cavity test bar die.

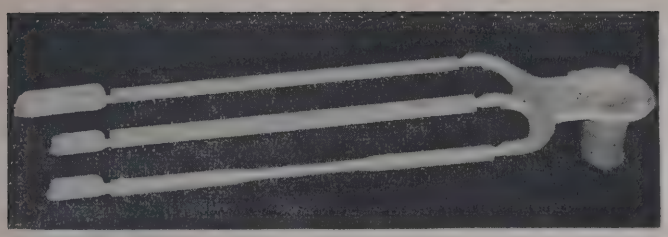


Figure 5.—Ejector view of test bar die casting.

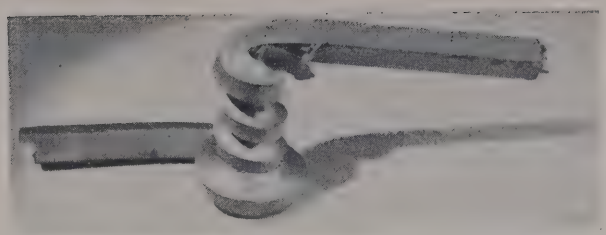


Figure 9.—Waster bar bent while hot in attempt to cause cracking.

Experimental Programme

In the present work, only the *ASTM No.* 23 (Zamak 3-type) copper-free alloy was investigated since it is the most used of the standard zinc die casting alloys, accounting for more than three-quarters of all zinc alloy die castings. This alloy has high dimensional stability combined with excellent retention of impact properties. *ASTM* alloy No. 25 (Zamak 5-type, having 0.75 to 1.25 per cent Cu) is somewhat stronger and has better castability, but impact strength and dimensional stability drop off when the alloy is used at elevated temperatures.

The following ranges of impurities were investigated: lead, from 0.001+ to 0.015 per cent in seven levels; cadmium from 0.001 to 0.010 per cent in three levels; and tin in two levels, one below 0.001 per cent (which is representative of normal tin content of 99.99 grade zinc) and one at 0.003+ per cent (which might be considered as representative of the tin content of re-melted zinc alloy scrap which has become contaminated during the alloying and re-melting operations). Other elements were kept constant at the following levels:

Al	4.1 per cent
Mg	0.04+ "
Fe	0.01 "
Cu	0.01 "

A limited number of compositions were tested, with copper at the upper *ASTM* specification limit of 0.1 per cent.

All materials used in the experimental programme were supplied by The Consolidated Mining and Smelting Company of Canada, Limited. The main alloy compositions (totalling about 50,000 lb. of metal) were prepared at Trail, B.C., and further alloying additions were made at Ottawa.

Experimental work was started in October, 1949, and completed in September, 1950. In all, over 100 compositions were cast, and for each composition approximately 200 die castings were produced for testing after having discarded at least fifty preliminary die castings in each case in order to stabilize the temperature of the die. The casting machine was operated at about 150 shots per hour, the individual castings being produced with a cycle time of: die closed, 10 sec.; shot time, 7 sec.; total cycle time, 15 sec. At every tenth shot, the die was dressed with *'Die Slick'* No. 11, temperatures were checked, and samples taken where necessary. The required number of test bars was determined by statistical analysis on early melts, and these bars were selected at random from the 200 die castings, as indicated in Table 1.

Table I

	Tensile	Impact	Stability
Aged 10 days in air	5 specimens	20 specimens	5 specimens
Aged 10 days in air at 95°C.	5 specimens	10 specimens	5 specimens
Aged 10 days in steam at 95°C.	5 specimens	10 specimens	5 specimens
Tested at 0°C. after ageing 10 days in air		20 impact bars	
Tested at −10°C. after ageing 10 days in air		10 impact bars	
Tested at −20°C. after ageing 10 days in air		10 impact bars	
Tested at −40°C. after ageing 10 days in air		10 impact bars	

The 'as cast' bars for ageing at room temperature and in air at 95°C. were lightly ground along the parting line with a fine sand wheel to remove flash. Bars to be aged in steam were de-greased before insertion in the steam cabinet, where it was found that any fine flash corroded away in the course of the 10 days' exposure.

The tensile specimens were broken in a Baldwin-Southwark, Tate-Emery testing machine, using wedge grips and a uniform cross-head speed of 0.40 inches per minute. Percentage elongation was measured by placing the fractured ends together and measuring, by means of micrometer calipers, the distance between punch marks originally two inches apart.

Impact tests were carried out on unnotched three-inch specimens using a Tinius-Olsen machine set to a 120 ft.-lb. capacity. Some of the low-temperature impact tests were carried out on a smaller machine of 30 ft.-lb. capacity for greater sensitivity.

Dimensional stability measurements were made on impact bars which had previously been cut to 6 inches in length prior to ageing. All measurements, before and after ageing, were made in a constant-temperature room (temperature 25°C.) using a micrometer gauge, which allowed measurements to be made with an accuracy of ± 0.0001 inches.

Figure 6 shows the cabinet used for ageing specimens in steam at 95°C. This is a standard rectangular cabinet, conforming to former *ASTM* requirements, and patterned after a New Jersey Zinc Company design (6).

Other tests included radiographic examination of selected bars from each composition, density determinations, and metallographic examinations. The surface appearance of the bars after steam ageing also provided an important guide to the quality of the die casting.

Discussion of Results

Before presenting the test results, mention should be made of some of the factors which affect the quality of zinc die castings, and which could, if not properly controlled, overshadow the effect of intentional variables of the research programme. Some of these factors are metal and die temperatures, pressure, and speed of injection, uniformity of temperature throughout the die cavity, and die lubrication. It is well known in the die casting industry that, to obtain the best results, these factors have to be checked and established for each new or corrected die, and that there are only very general rules, gained through years of shop experience, which make it possible to speed-up the breaking-in period of new dies. The same problem was encountered in the present

investigation, although experience from a previous research project was of considerable help.

Casting Variables

Occasionally, very erratic results were obtained and, in an attempt to find the cause, a number of tests were carried out. Unfortunately, only a limited time was available for such tests, and none of the factors tried appeared to have any decisive influence on the mechanical properties obtained from the castings. However, as the experience gained during this phase of the work helped in the production of sounder test bars with a narrower spread of properties, a brief description of these tests is given below.

Injection pressures on the metal in the gooseneck were varied from 2,100 p.s.i. to 1,300 p.s.i. at injection speeds of medium and fast. In general, slightly higher properties were obtained at the highest pressure, although the shock on the machine was excessive. The highest injection speed appeared to consolidate the metal at the vent end, giving greater scatter of results from gate, middle, and vent. Consequently, a medium-high injection speed and an intermediate pressure (1,700 p.s.i. on metal in gooseneck, 800 p.s.i. pump pressure) were chosen as standard.

Dies were normally operated at a temperature of about 210°C. (410° F.), and experiments on higher and lower die temperatures failed to give results significantly different from the standard. There was some indication that low die temperature gave higher results, but this was counteracted by increased trouble in operation due to freeze-up in the sprue, which would lead to increased scatter of results.

Inconclusive results were also obtained with experiments on increasing and decreasing the heat on the nozzle. It might be mentioned that it is thought that accurate control of nozzle temperature by means of a controlled resistance heating element might reduce the scatter of results. In all casts, the nozzle was heated with a gas burner and the temperature was judged by appearance.

Varying the water on the sprue had no effect on the properties obtained from the castings but did affect the ease of working of the die casting machine.

Due to the failure of a controller, the metal became overheated on one occasion to over 500°C (930°F.). Test bars from the overheated metal gave higher results, especially after steam treatment, than those obtained for the same metal cast at ordinary temperatures. This is contrary to what might be expected because of oxidation of the metal and slower cooling in the die. It might be mentioned that the melting and casting conditions in this investigation were much less turbulent than in normal industrial practice.

On the early casts, scrap sprues, etc., were returned direct to the melting pot, and erratic results were obtained, which, from radiographic inspection, appeared to be associated with porosity and dross inclusions. The technique was therefore changed and all scrap and replacement ingot metal was charged into a separate melting pot which was maintained at an even temperature and skimmed regularly. This was then ladled into the melting pot of the die casting machine as required. This procedure caused a considerable improvement in soundness, mechanical properties, and reproducibility of results of the test bars.

It was also found that more reproducible results between different casts were obtained by keeping a constant volume of nitrogen in the accumulator.

Impact Tests

The greatest variations in results were obtained in the impact test. As previously noted, this test is carried out on unnotched specimens of approximately 0.25 - inch - square cross-section, and 3 inches in length. Under some conditions of testing, the gate section of the impact bar showed decidedly inferior properties. The differences between gate and vent ends are thought to be due to variations in the freezing rate and the mode of flow of the metal in the two ends of the die cavity. These differences may be exaggerated by the use of a 9-inch impact bar instead of the 6-inch length used by other workers in this field. Improvement in casting technique resulted in higher impact values generally, and minimized differences between gate and vent end impact values of bars aged at room temperature and in dry air at 95°C.

Considerable scatter of impact results was obtained in tests carried out at 0°C. and −10°C. In some cases, adjacent bars varied from 2 to 3 ft.-lb. to more than 50 ft.-lb. in impact strength. This peculiarity is readily explained since the impact-temperature curve is almost vertical at just less than 0°C. (19). A study of the effect of impurity additions on impact properties in this temperature range would have to be carried out under much better controlled conditions than are generally available on a commercial-size die casting machine.

All test results were subjected to statistical analysis, with the exception of impact results on specimens aged at room temperature and at 95°C. in dry air. In both these instances, a number of the specimens did not break in the impact test but bent and were pulled through the anvil. The data were, therefore, not amenable to statistical treatment.

Statistical Design and Analysis

A factorial design was used with twofold replication of compositions over the whole experimental design and an additional replication over the portion of the design of most interest, *i.e.*, from 0.003 to 0.007 per cent Pb, and 0.001 to 0.005 per cent Cd. Compositions were cast as nearly as possible in random order, although complete randomization was not achieved because a certain degree of order had to be preserved to conserve the limited supply of alloy.

After about one-sixth of the design was completed, a preliminary analysis indicated that more bars were being tested than warranted by the appreciable variation between replicate melts, and the number of bars tested per melt was reduced considerably, to give the final test programme as outlined under the heading *Experimental Programme.*

The complete data were treated by analysis of variance. Since a large number of tests of significance were to be made, and because melts were not made in completely random order, a critical region of small size was used ($\alpha = 0.01$).

In addition, the power (20) of each test was determined in order that an estimate could be made of the maximum effect likely to escape detection. Again a critical region of small size was chosen ($\beta = 0.01$).

With this treatment we would expect to claim a significant effect where none actually existed about once in a hundred times and we have estimated the magnitude of change in physical properties which escape detection only once in a hundred.

Dry Ageing

Apart from a slightly lower elongation value for compositions containing 0.007 per cent Pb or higher, which amounted to about a 1.5 per cent decrease from an average elong-

ation value of 17.6 per cent, there were no significant differences in any of the mechanical properties between any of the various compositions tested after ten days' ageing at room temperature or at 95°C. in dry air. Average properties of all compositions tested under these conditions are shown in Table II. Results compare favourably with those published by other investigators.

TABLE II

	Specimens Aged 10 Days at R.T.	Aged 10 Days at 95°C. in Dry Air
Tensile strength	41,655 p.s.i.	34,805 p.s.i.
Elongation	17.6%	23.1%
Impact strength	46.0 ft.lb.*	45.7 ft.lb.*
Measured at −20°C.	4.2 ft.lb.	—
" −40°C.	2.5 ft.lb.	—
Net dimensional change in 6-inch bar	−0.0015 in.	−0.0031 in.
Specific gravity of castings	6.63†	

*Average of gate, middle, and vent sections.
†Average of a large number of specific gravity determinations over the range of compositions tested.

Steam Ageing

The effect of impurities on the mechanical properties of the alloys is most pronounced in specimens aged for ten days at 95°C. in the steam cabinet. The 10-day steam test is a standard test developed for evaluating zinc die casting alloys and, while subject to the limitations of an accelerated test, may be considered to approximate the performance of zinc alloys aged under normal conditions for an extended period. The exact mechanism of intergranular corrosion observed in zinc die casting alloys has received considerable attention, as already discussed.

It is well known that, even when the highest purity metal is used for alloying purposes, the properties of zinc-base alloys are materially reduced when aged under these conditions. In general, tensile strength drops from 41,000 p.s.i. to 34,000 p.s.i., and impact strength is lowered by a variable amount, depending on the conditions under which the alloy is cast. As for dimensional stability, there is a small expansion for *ASTM* alloy No. 23, which does not, however, offset the normal shrinkage of the alloy when aged at elevated temperatures.

The general effect of impurities on the mechanical properties of *ASTM* alloy No. 23 after ten days' ageing in steam at 95°C. is shown graphically in Figures 7 and 8. These figures are based on a large number of determinations and on the statistical analysis of these results. They illustrate trends in the behaviour of tensile, elongation, impact, and dimensional stability properties related to variations in lead, tin, and cadmium contents. In Figure 7, results are presented for alloys containing less than 0.001 per cent Sn; in Figure 8, for alloys containing 0.003+% Sn. From an examination of Figures 7 and 8, and from the detailed statistical analysis, the following conclusions can be drawn:

(*a*) *Tensile Strength*: In the tin-free alloys, lead below 0.012% caused only slight reduction in properties. With lead in the order of

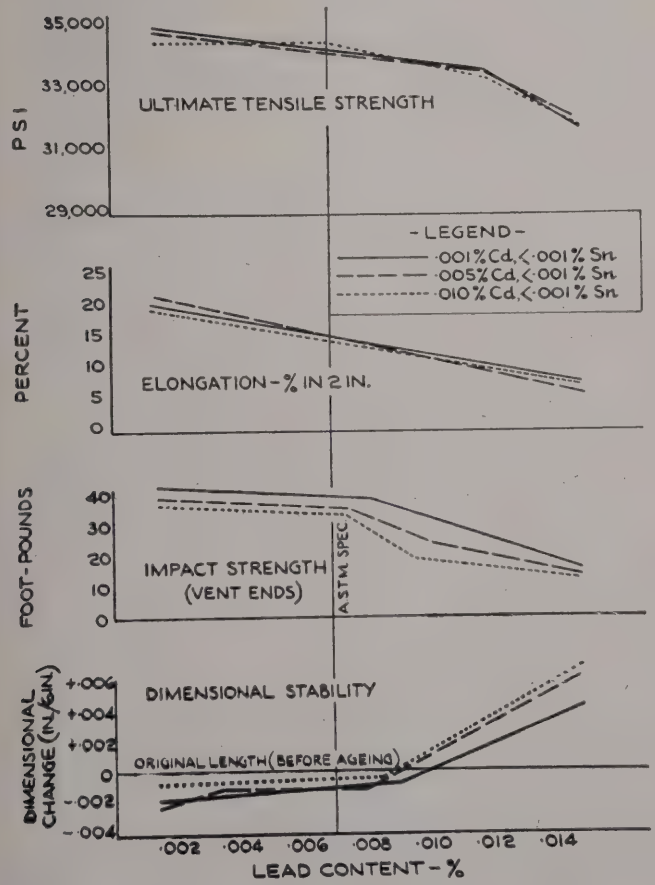


Figure 7.—Effect of lead and cadmium on properties of A.S.T.M. Alloy No. 23.

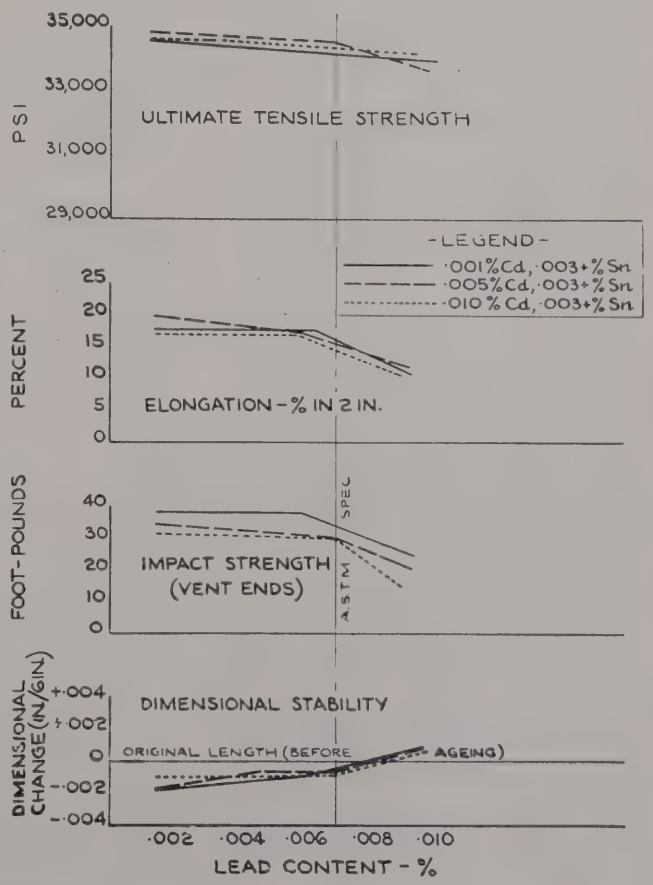


Figure 8.—Effect of lead, cadmium, and tin on properties of A.S.T.M. Alloy No. 23.

0.012% and higher, fairly rapid deterioration commenced.

The presence of 0.003+% tin in the alloy gave a similar curve, but in this case the highest lead content tested was 0.009%. Under these conditions, the presence of this amount of tin had no significant effect.

Cadmium in the composition ranges tested (up to 0.010%) had no effect on tensile strength.

(*b*) *Elongation*: With the tin-free alloy, the elongation dropped steadily with increasing lead content, until at 0.012% lead the elongation value had reached 10 per cent. At the *ASTM* specification limit of 0.007% lead, elongation values were satisfactory, averaging approximately 15 per cent.

The presence of 0.003+% tin appeared to have a slight detrimental effect on elongation, particularly at the lower levels of lead (below 0.005% lead).

Again cadmium had no significant effect.

(*c*) *Charpy Impact Strength*: Results shown in Figures 7 and 8 for impact strength are vent end values only. Similar trends were obtained for middle and gate end values, although impact strengths were generally lower for middle sections, and considerably lower for gate sections, for all compositions tested, and the scatter of results increased.

From Figure 7 (less than 0.001% tin content), it is evident that there was a pronounced drop in impact strength between 0.007 and 0.009% lead. Reference to Figure 8 shows that the presence of 0.003+% tin lowered the impact values for all levels of lead and cadmium by about 4 ft-lb.

In both cases (*i.e.*, both with and without tin present), cadmium had a significant detrimental effect.

(*d*) *Dimensional Stability*: In Figures 7 and 8, the net dimensional change (in inches) occurring in the 6-inch stability bars during the 10-day steam ageing period is plotted against the lead content of the alloys. At the higher lead levels, all curves show a net expansion, indicating that growth of the specimens due to intergranular corrosion has been greater than the normal shrinkage of the specimens. In the absence of tin (Figure 7), there was no significant difference between lead levels of 0.001+% and 0.009%, but at higher levels of lead, *e.g.*, 0.012% and up, a marked increase in growth was shown, indicating rapid deterioration of the alloy.

It was found (Figure 8) that the presence of 0.003+% tin had a small detrimental effect. The amount of shrinkage at the lower lead levels was decreased and the expansion at the higher lead level (0.009% Pb) increased. Cadmium up to 0.010% had no significant effect.

Thus, summarizing the effect of impurities on Zamak 3-type alloy, lead contents of 0.012% and higher caused a serious drop in tensile and elongation properties. Neither tin (at 0.003+%) nor cadmium (up to 0.010%) had a really significant effect on these properties. Lead caused a drop in impact strength between 0.007% and 0.009% lead content, and cadmium (0.005 and 0.010%) and tin (0.003+%) each showed small detrimental effects on this property over the entire range of lead contents tested. Dimensional stability was affected by lead, particularly by lead contents greater than 0.009%, and to a lesser extent by tin; cadmium (up to 0.010%) had no significant influence on dimensional stability.

From these results it would appear that satisfactory die castings can be produced from alloy containing *ASTM* limits for lead and cadmium (0.007% and 0.005% respectively), and that the presence of 0.003+% tin is detrimental in some respects. From the standpoint of its effect on mechanical properties, cadmium appears to have the least effect of the metals investigated, and amounts up to 0.010% could be tolerated. It is considered that any further increase in lead and tin above 0.007% and 0.003+%, respectively, will bring the alloy close to the point where more rapid deterioration of the alloy in moist atmospheres can be expected.

In all this work there has been a fairly large variation between replicate tests, so that a large programme was required to establish the results presented. This emphasizes the necessity of having proper casting conditions as well as adequate metal purity in the production of quality die castings.

Special Tests

In conjunction with the casting, a number of investigations were carried out to determine the cause of the wide variations sometimes experienced between adjacent bars of the same melt or bars of the composition cast from different melts, and for other purposes.

Density measurements were carried out on a large number of sections in an attempt to correlate impact properties with the soundness of the bar. The results obtained showed that there was no correlation between impact strength and density, although bars with high density usually did have high impact strengths. From the results obtained, it was suggested that the most important factor affecting the impact strength is the flow of metal and rate of cooling affecting the structure of the portion of the bar struck by the impact hammer.

Numerous bars were radiographed, and, again, no correlation could be obtained between the apparent soundness of the bars and their behaviour in the impact test.

It was thought that surface preparation of bars before insertion in the steam cabinet might affect the results, but a number of bars prepared in different ways showed no significant difference in impact properties after steam exposure. In this connection, a number of impact tests were made on bars which were machined with different designs of notches. These tests indicated that the steam ageing treatment had the effect of notching the surface of the test bars in an erratic manner, depending on the location of the corrosion penetration. This, in turn, gave widely different impact properties depending on the location of the 'corrosion notch' with respect to the striking face of the impact tup.

Metallographic examination of a large number of samples was also carried out to determine the cause of the variable impact properties. This phase of the work was for the most part concerned with examination of polished sections taken from specimens after ageing for ten days in the steam cabinet. No definite correlation between structure and impact properties was noted. Depth and extent of intergranular corrosion was very erratic, even for bars of like composition, and for the highest purity metals tested. However, when the lead content exceeded *ASTM* specifications, the depth and extent of corrosion became increasingly severe.

In view of the relative insensitivity of the tests carried out, a small jig was designed to test the centre waster bar for hot shortness. The aim was to bend this bar through 180° as it came hot from the die. However, at this temperature all compositions had high ductility, as shown by Figure 9.

In addition, an apparatus was constructed which, according to Sundwick (21), would give a meas-

ure of the fluidity of the alloys. However, experiments with this apparatus failed to give reproducible results and the experiments were discontinued.

Effect of Copper

Additional to the main programme which was based on the copper-free *ASTM* No. 23 alloy, a limited number of tests were carried out on alloys with copper added to the maximum limit allowed in *ASTM* specifications, *i.e.*, 0.1 per cent. The objective of this study was to ascertain whether or not this amount of copper would change the pattern of results obtained for the alloy when substantially copper-free. This was done since many die casters operate with alloy containing copper to this upper limit due to contamination. The results of these additional tests were somewhat erratic but showed that, in general, the addition of 0.1% copper is beneficial in that the impurity range appears to be less critical than in the copper-free alloys.

Effect of Indium

As a side issue to the main programme, a small-scale investigation was undertaken on the effect of indium contamination in zinc die casting alloys. No data were available in the literature and it was decided to determine the effect of small concentrations of indium which might be introduced as a stray element should indium find application, for example, in impervious electrolytic protective coatings on zinc alloys.

Figure 10 shows the results obtained after ageing for ten days in the steam cabinet. The graphs reveal that, with less than 0.001% In, most properties have been seriously affected. In somewhat higher concentrations, *e.g.*, 0.01%, indium causes brittleness and considerable lowering of properties even in the absence of heat or moisture. The alloys were noticeably hot short at 0.01% In.

Figure 11 shows photographs of steam-aged specimens with increasing indium contents. The increasing depth of the intergranular corrosion is plainly visible. A few tests were carried out with indium as high as 0.05%. After ageing in steam these specimens were found to be broken or too badly distorted to be tested.

At 0.004% indium, 0.07% copper was added. From the surface appearance of the bars, this did have a beneficial effect in reducing the extent of corrosion, but by this time the properties were so bad that the copper addition caused no significant improvement.

Thus, indium is by far the most detrimental of the impurity metals investigated in this programme. Fortunately, it is not normally present in refined zinc or in zinc die casting alloys. Care should be exercised to ensure against it entering zinc die casting alloys with alloying elements or as a tramp metal.

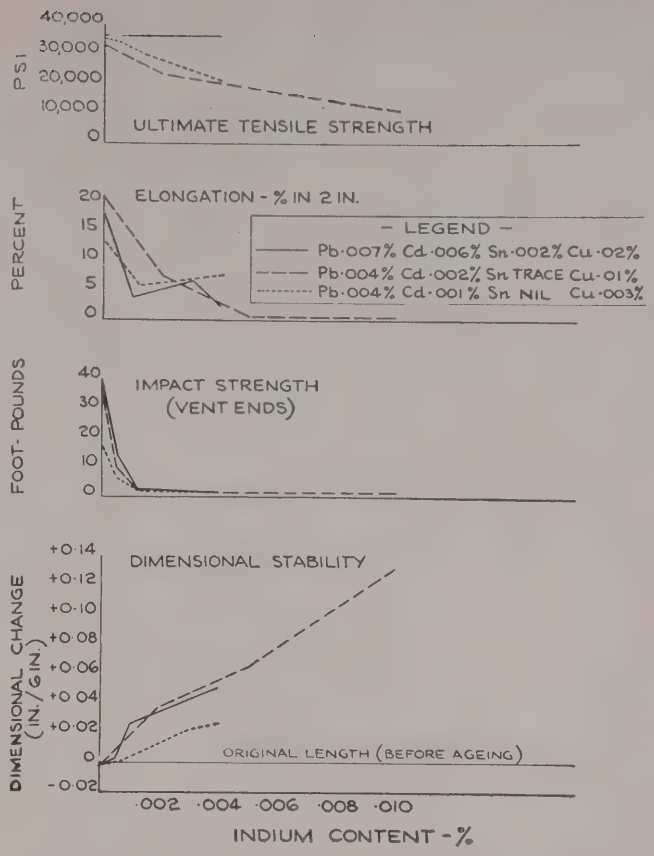


Figure 10.—Effect of indium on properties of A.S.T.M. Alloy No. 23.

Summary

A series of alloys were die cast to determine the effect of impurities on the properties. The alloy was basically Zamak-3 (*ASTM 23*) to which was added 0.001 to 0.015% lead, 0.001 to 0.010% cadmium, and *nil* to 0.003+% tin, and combinations of these. The effects of additions of 0.1% copper and up to 0.05% indium were also investigated.

The results show that, under conditions of steam ageing, lead affects all properties, being seriously detrimental to tensile strength and elongation at about 0.012% lead, and to impact strength and dimensional stability at about 0.009%. Tin at 0.003+% exerts a detrimental effect on impact strength and on dimensional stability. Cadmium up to 0.010% has a small detrimental effect on impact strength only.

Alloys with a copper content of 0.1% appear to be less sensitive to the effect of impurities.

Indium, even in amounts as low as 0.001%, causes severe deterioration in properties and corrosion resistance.

A number of special tests, carried out to determine the cause of the erratic impact results sometimes encountered, met with little success but suggested that the surface condition of the bar where struck by the impact tup was the most important factor.

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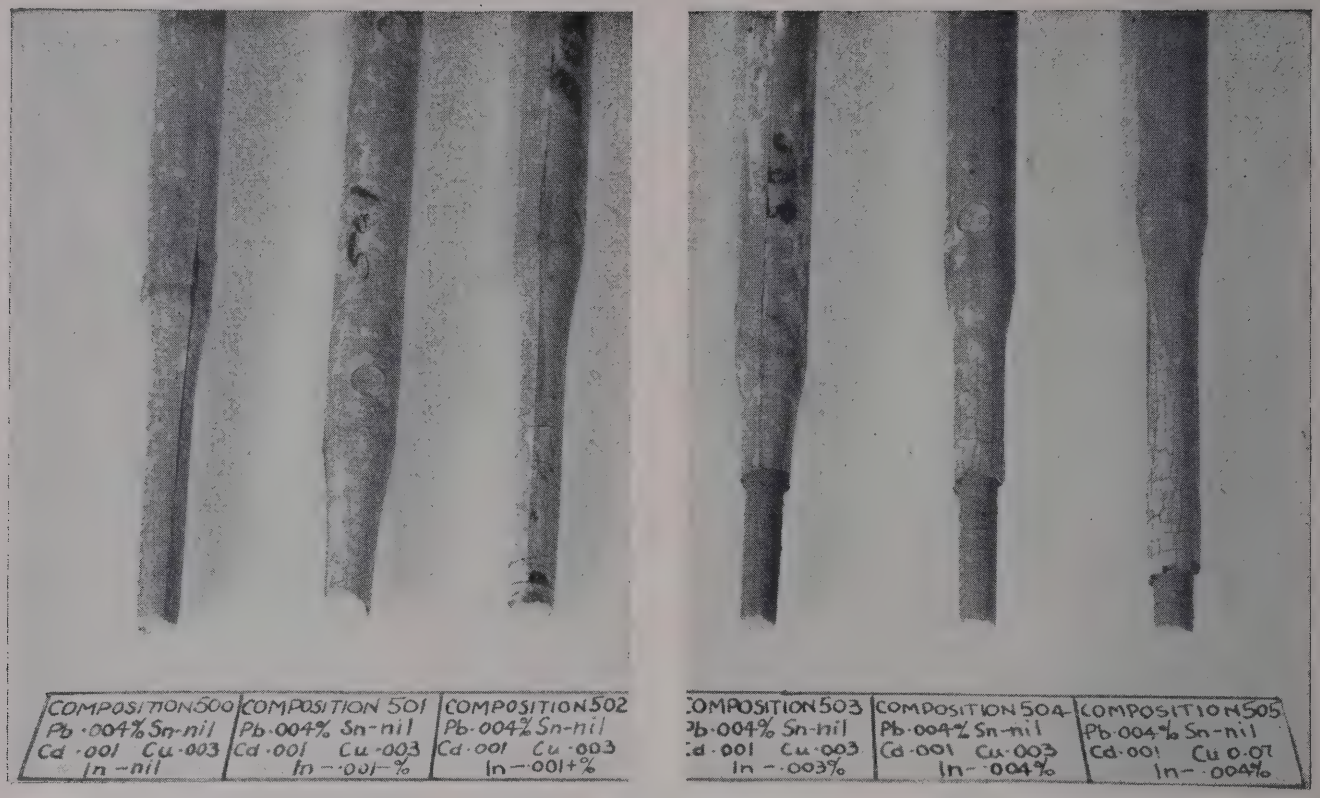

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* * *

DISCUSSION

MR. W. M. PEIRCE (*New Jersey Zinc Co., Palmerton, Pa.; President, AIME*): Even though I am a visitor, I would like to make one or two comments. The first job I had with the New Jersey Zinc Company was developing zinc die casting alloys, and the alloys under discussion here were developed as a result of that programme, so it was very interesting to me to hear this subject. There are two points that I think are worthy of comment. Aside from the mechanical properties of the specimens as they have been tested here, there is another very important effect of some of the impurities which were mentioned, especially of tin, cadmium, and indium, but in the case of cadmium I would like to comment on it particularly. You have to be able to die cast these alloys and, while cadmium does not show up as quite such a harmful impurity from the standpoint of the mechanical properties after you get the piece die cast, it does make the alloys hot-short and makes them very difficult to die cast. Indium has the same effect on zinc alloys.

There is one other point that I think should be commented on, namely, that the effect of steam treatment on impact strength must be analyzed a little bit. These alloys are very sensitive to the notch effect, and you can take a specimen that has been corroded on the surface and has shown a very substantial drop in impact strength and, by polishing it and restoring a smooth surface, you can bring back the impact strength, so that the effect that is noted in the steam test on impact strength is really a combination of two factors, the actual penetration and then the exaggerated effects of the roughening of the surface.

MR. BELL: I would like to ask Mr. Peirce at what level of cadmium the alloys, such as we are discussing, would become hot-short in practical die casting.

MR. PEIRCE: I expect it is nearly a straight-line function. You have some hot shortness obviously in any alloy but here it increases almost in a straight-line function with a very low content.

AUTHOR'S REPLY: We are indebted to Mr. Peirce for his comments on the paper. With regard to hot shortness, we appreciate that our test bar castings are much simpler than the complex die castings produced industrially and are less likely to show up such effects as hot shortness. However, as mentioned, when indium was present, hot shortness became apparent by the bars falling away from the gates on extraction. No such effect was observed with cadmium even at 0.010 per cent.

The effect of steam treatment on the surface of the bars, and the related notch-sensitivity of the alloys, are worthy of further study. It was interesting to hear that Mr. Peirce's long experience confirmed our tentative suggestions of the mode of failure of the steam-treated bars in the impact test.

Water — An Industrial Raw Material *

By J.F.J. THOMAS †

(*Ottawa General Meeting, January,* 1952)

(*Transactions, Volume LV.* 1952, pp. 262-269)

ABSTRACT

Both the quantity and quality of waters available as raw materials are important to Canadian industry and to the national economy. The huge demands of many Canadian industries is illustrated by tabulated figures, based on the 1950 Canadian production of certain products, and on United States figures for the gallons of water required to produce a unit quantity of each product. It is suggested from comparison with American survey results and figures that Canadian industrial water consumption is 1 to 1½ billion Imperial gallons per day. Since present information on Canadian usage is meagre and it is not certain that American survey results can be directly applied to Canadian industry, this estimate must await confirmation in future studies.

The hydrologic cycle and the principal factors affecting the quantity and quality of water, namely, climate and season, topography, geology, vegetation, and human activities, are discussed and the principal salts dissolved in water which affect industrial use are enumerated.

The present survey studies by the Mines Branch on water quality are outlined. Besides chemical analyses of municipal water supplies within a watershed, periodic sampling and analyses of major surface waters for a one-year period is being carried out. The importance of studies on seasonal variation in quality is stressed by comparing the dissolved salts in various river waters with river flow. Figures showing such relationships in the Rideau river and Ottawa river at Ottawa, the Columbia river at Trail, B.C., and the Battle river at Unwin, Sask., are presented. Different rivers show different quality changes with changing flow, and possibly only after long-term studies can the relationships be predicted.

Quality studies on water may be as important as studies on many other raw materials. The quantity of the raw material, water, is not inexhaustible nor is the quality indestructible, and more realization of its importance, followed by conservation and intelligent use, is desirable.

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Introduction

TO ANYONE interested in the specialized field of water quality, water treatment, and industrial water use, it is soon evident that the public and, surprisingly, a fair percentage of industry, have only a limited appreciation of the importance of water to the national economy. Although there is a growing realization of the serious effect on health and recreation of any deterioration of national water resources, there is still considerable ignorance regarding the importance of water as an industrial mineral, or raw material. Far too frequently the questions are asked, "why are you analyzing water?", and "why is the chemical quality important?". In the following remarks an attempt is made to answer these questions, show the national importance of water, and outline the present and proposed activities of the Industrial Minerals Division of the Mines Branch in this field. Since the Division's studies are far from complete and are being published as quickly as possible in the form of Mines Branch Reports, it is not intended to anticipate these reports in any detail. Rather, this discussion will be of a general nature, drawing largely on recent technical literature for illustrative material.

Water Quantity

The dependence of all life on water is common knowledge but the importance of the availability or quantity, and of the suitability or quality, of the water is not so well recognized. History records the disappearance, decline, or change in entire civilizations partly because of a lack of, or an excess of, suitable water supplies. Without suitable water in sufficient quantity, many other natural resources could not have been developed to the extent they now are in many parts of the world.

In certain areas of the Canadian West, the lack of water has caused depopulation, limited the economy to almost one type of agriculture, and led to considerable hardship. This has necessitated the development of huge and costly irrigation projects. In other sections of Canada, excess water has caused widespread suffering and monetary loss, such as in the recent Fraser River and Red River floods. Without sufficient water, Canada would not have the huge hydro-electric developments so vital to her industrial growth.

Canada and the United States are blessed with more than their share of the world's fresh-water supply. It is reported that the United States has three-quarters of the world's supply within or on its borders. Six per cent of Canada's land area is said to be covered with fresh water, and we have some of the world's larger rivers — the MacKenzie, 2,514 miles from Great Slave lake to the Arctic Ocean; the Nelson River system, including the North and South Saskatchewan rivers, draining most of the prairies from the Rocky mountains to Hudson bay; and, of course, the great St. Lawrence River-Great Lakes system.

Except in certain western areas, Canada is fortunately not seriously restricted as to a supply of water, although in some other areas, particularly southern Ontario, future industrial growth is already being limited by this factor.

Quality of Water

In other sections of the country, while sufficient water is available, the quality of this water is far from satisfactory. An extreme case of the importance of quality is the necessity for fresh water when one is surrounded by sea water or by alkali water such as is found on the western plains. Quality of water is generally associated with potability, that is, whether the water is bacteriologically pure and suitable for drinking. While potability is important to industry, quality as discussed here refers to the amount and nature of various chemical materials dissolved or suspended in the water. To industrial users, these are

often more significant than the question of potability.

Many factors affect the quality of water, and different industries demand a different quality in the water when used as a raw material. Even when water is used only as a means of conveyance, poor quality may give rise to corrosion with resultant costly breakdowns and necessity for repairs.

If the quality available is not satisfactory, industry must treat the water or else secure a more satisfactory supply. Treatment costs may in some cases become prohibitive if initial quality is poor and the demand high, even though the raw water may be 'cheaper than dirt'. It is gratifying to find more and more industrialists realizing that it is often good economy to secure a superior quality of water rather than purchase higher grades of other raw materials in order to market a higher grade product.

A knowledge of the quality of waters now available in Canada is therefore desirable because:

(1) Being a major natural resource, the amount and nature of it should be known.

(2) Information on quality will assist industry in the location of new plants, in the expansion of present plants, and in determining the treatment necessary to adapt a water to a particular end use.

(3) Unless present water quality is known, any deterioration due to natural and human causes is difficult to assess. Quality information is a red light warning of the need for conservation and more careful waste disposal.

(4) More knowledge on water quality and related properties of water increases the possibility of improving treatment methods, and of solving major problems arising from the industrial use of water.

Factors Affecting Water Quantity and Quality

The hydrologic cycle, shown in Figure 1, maintains the amount of water in the world relatively constant and controls its distribution availability, and quality. Ocean water, evaporated by the wind and sun, is carried by winds as clouds, spray, etc., over the land, where it is precipitated as snow, rain, or hail. Some of this water is retained in the top soil and is re-evaporated or used by vegetation. A portion penetrates more deeply into the ground and is stored in porous rock, sand, or gravel. A portion runs off as streams and rivers, eventually returning to the ocean. The ground water is continually being replenished, since it is lost as springs and artesian wells, many of which may enter directly into the streams and rivers. Water which is retained on the surface as lakes, ponds, marshes, glaciers, and snow, partly is re-evaporated and partly feeds the streams and underground water reservoir. The U.S. Geological Survey estimates that, on the average, 21.5 inches of a 30-inch rainfall is re-evaporated or used by plants; the remaining 8.5 inches runs-off into streams or into the ground water table. It is this 8.5 inches which provides the water used by industry.

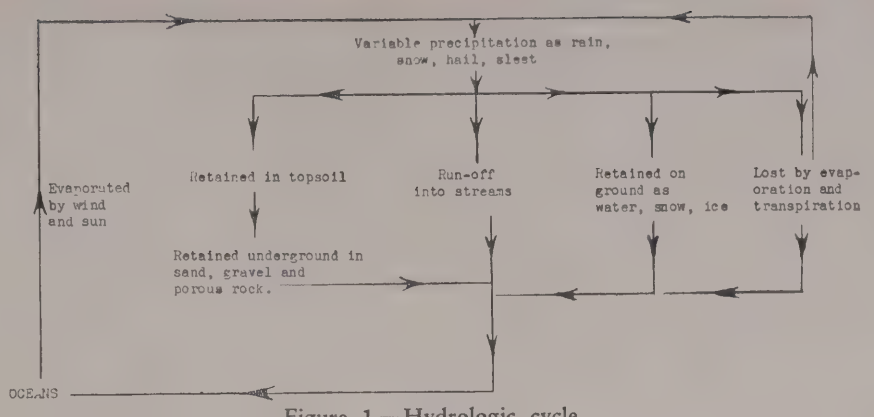

Variable precipitation as rain, snow, hail, sleet
Evaporated by wind and sun
Retained in topsoil
Run-off into streams
Retained on ground as water, snow, ice
Lost by evaporation and transpiration
Retained underground in sand, gravel and porous rock.
OCEANS

Figure 1.—Hydrologic cycle.

The distribution of the water on the land, and its quality, depend upon many factors, the principal being climate and season, topography, geology, vegetation, and human activities.

Climate and Season

Everyone is familiar with the effect of climate and season on the amount and nature of precipitation. They affect quality primarily from the amount of precipitation and there is little man can do to control their effect.

Topography

Topography, that is, the surface relief, elevation above sea level, slope of land, and stream pattern, is most important since it not only influences the amount of precipitation but also determines the rate of run-off of the precipitated water. Rain, when falling, picks up dust and gases, especially carbon dioxide, which assist in the solution of chemicals from the soil. If the area is heavily wooded and the elevation low with only slight slope, the run-off is slow and the water is in contact for some time with soluble material. Thus, topography determines, by stream pattern, the nature of the soil which the water contacts and, by elevation, slope, etc., its length of contact and so, to a large extent, the total pick-up by the water.

Geology

Geology of the land determines largely the extent and nature of the material dissolved by the water. If the soil is relatively insoluble and the run-off rapid, little solution may occur. In some areas, the geology is such as to permit excessive penetration into the ground, where long contact causes considerable solution of material. If water is in contact with limestone, gypsum, or dolomite, it will pick up these materials; if in contact with land in which alkalis predominate, the water will have a high percentage of salts such as sodium sulphate and sodium bicarbonate.

Vegetation

Vegetation on the land affects both the quantity and quality of the water. The vegetation retains much of the water, releasing it but slowly and thus preventing too rapid a run-off. However, this longer contact with vegetation results in considerable solution of colouring matter, producing the highly-coloured waters found in swamps and in wooded northern areas.

Human Activities

Human activities within a watershed can have most marked effects on water quality. By seriously disturbing the natural availability by deforestation, damming of rivers, construction, farming, etc., the length of contact of the water with the soil, and even the type of soil contact, can be radically altered. Man-built dams may hold back normal run-off, and deforestation may cause excessive run-off followed by drought. These conditions are becoming far too common in Canada. In southwestern Ontario, denudation of the land has caused serious flooding followed by drought, and once permanent streams are now as much as 85 per cent dried up during part

of the year. It is said that, in this area, there remains a lower percentage of wooded land than in the old state of Germany.

Excessive pumping of ground water can seriously affect water quality since a less suitable water may be drawn into the water table. Along the southern seaboard in the United States, on Long Island, and in parts of Texas and California, over-pumpage has caused the sea to encroach upon the fresh-water table. Excessive pumping in parts of Texas has lowered land elevations two feet. In some areas of California, the elevation has been lowered five feet within twenty years. In the area around Baltimore, Maryland, it is now necessary to go 146 feet deeper than in 1916 in order to get water.

Human carelessness in the disposal of trade wastes and sewage may ruin the quality of available water supplies for many uses. The waters of some rivers on this continent have been so badly deteriorated in quality that they are no longer fit for cattle to drink. Over 22 million dollars has been appropriated to be used over a five-year period in reclaiming the quality of the water in one American watershed. In a recent report the International Boundary Commission states that the Detroit river is receiving daily about 6½ tons of phenol, 4.3 tons of cyanide, 9 tons of waste oil, and immense quantities of sewage and other contaminants.

It is very easy for man to adversely affect or completely ruin a good quality water provided by nature. It is not so simple to restore the quality of, or to improve, a water naturally heavily mineralized.

Industrial Demands

The importance of water to industry is perhaps best illustrated by indicating some of the demands regarding quantity and quality. Table I shows some of industry's requirements regarding quantity. The unit quantities of water, converted to Imperial gallons, used in the preparation of this Table are those reported by S. T. Powell, H. E. Jordan, and others, based upon United States surveys. It will be noted that the amount required to produce several basic products is enormous. Roy Green has also reported that, in certain industries, 1,400 U.S. gals. of water are required for every dollar of product value added, while in other industries only 3 to 4 U.S. gals. may be needed. Using Canadian figures for the production of certain products in 1950, and these unit water quantities, the amount of water used per year and per day for this portion of Canadian production has been estimated. The breakdown of annual production is not always similar to the breakdown used in determining the unit water quantity; for example, milk products are reported to use from 0.09 to 0.21 gals. per lb., but no information is given as to which milk product uses which unit amount. Many important industries, such as the textile industry, are not shown in the Table, nor is the huge demand for water by agriculture and for hydro-electric power generation shown. It has been reported elsewhere that to produce one acre of corn requires 15 inches rainfall, or transpiration of about 3,000 tons of water, and that one loaf of bread requires 2 tons of water from seeding to the finished product. The quality of water is also of considerable importance to

Table I.—Estimated Partial Industrial Use of Water, 1950

Industry	Canadian Production		Estimated Water Use			Added Value
	Amount	Unit	Imp. gal./unit	m.g.yr.	m.g.d.	
Soap	230,731,000	lb.	0.21	48.45	0.13	$ 30,090,000
Beer	178,859,423	gal.	12.5	2,235.74	6.13	107,391,000
Whiskey	14,463,750	gal.	66.6	961.54	2.63	58,147,000
Manufactured ice (1949)	991,000	ton	213.3	211.38	0.58	5,392,000
Steel	3,383,575	ton	54,145	183,203.67	501.93	180,061,000
Cement	16,723,807	bbl	643	10,753.41	29.46	31,697,000
Paper	5,935,141	ton	32,500	192,892.10	528.47	
Paper board	876,894	ton	12,500	10,961.18	30.03	
Woodpulp:						
Groundwood	4,910,803	ton	4,165	20,453.49	56.03	636,241,000
Sulphate	1,053,588	ton	53,312	56,168.88	153.90	
Sulphite	2,221,773	ton	49,980	111,044.21	304.23	
Leather rawhide, tanned	3,892,000	lb.	6.66	25.92	0.07	18,135,000
Oil, refined	108,941,223	bbl.	641.4	69,874.90	191.44	
Gasoline	46,525,441	bbl.	297.4	13,836.67	37.91	130,338,000
Milk products:						
Cream	690,253,000	lb.				
Fluid milk	3,466,774,000	lb.	0.09—0.21	——	——	105,252,000
Butter	310,501,000	lb.				
Total milk	16,436,988,000	lb.				
Canning (1949)*	1,504,074,030	lb.	about 4.5	1,768.33	4.62	66,648,000
Aerated beverages	50,422,300	case	2.08	104.88	0.29	53,862,000
Total				674,544.75	1,847,.85	$1,318,002,000

*Fish products not included.

TABLE II.— WATER USAGE

INDUSTRY	NO. OF PLANTS REPORTING	ESTIMATED AVERAGE USE, g.p.d.	TOTAL WATER CONSUMPTION		AVERAGE PER CENT APPLICATION*		
			m.g.d.	m.g.yr.	Cooling	Processing	Boiler feed
Leather.....	30	110,600	3.32	1,211	Up to 10	80 to 89	7.1
Textile.....	112 { surface 89 / ground 23 }	384,400 / 8,000	34.21 / 0.18 } 34.4	12,555			
Pulp & paper..	67 { surface 63 / mixed 4 }	5,200 to 18,500	845.6	308,659	0 to 60	35 to 97	About 1.8
TOTALS.....	209		883.3 (0.88 b.g.d.)	322,425 (322 b.g.yr.)			

*Estimates not always based on total plants reporting.

agriculture since the consumptive use, that is, the amount of water transpired and evaporated by crops, varies with the quality. Too high a percentage of sodium or boron in the water is detrimental to the growth of many crops.

In the production of hydro-electric power, it is reported that about 67 gals. of water is required to produce a kilowatt-hour. Canadian figures for 1950 showed about 51 billion kilowatt-hours produced, that is, a total of about 3,420 billion Imp. gals. of water used. The United States is reported to use about 1,100 billion U.S. gals. daily to generate hydro-electric power. The quality of water is important to this industry primarily because of corrosion, scaling, and sliming problems.

It will be noted in Table I that the few Canadian industries represented have an estimated water use of 1.8 billion gals. daily. However, this figure probably includes a considerable amount of re-used water. K. A. MacKichan has recently estimated that, in the United States, the total 1950 consumption, including re-use, amounted to 173.6 billion U.S. gals. daily, of which 79 billion gals. was used for irrigation, 3.6 billion by rural users, and 77 billion or 44 per cent by private industry. About 13.6 billion gals. daily was supplied by municipal water-works, some, of course, to industry. Others have estimated that the United States uses only 100 to 150 b.g.d. (U.S.), of which about 25 billion gallons or 17 per cent is used by industry. On the basis of population ratios, Canadian total consumption would then be 10 to 11 b.g.d. (Imperial) and industrial use 1.6 to 1.8 b.g.d. Powell, Jordan, and others have prepared tables somewhat similar to Table I and have also determined water use in relation to the value added by manufacture, that is, the difference between the value of the finished product and the cost of the raw materials. In 1947, the amount of added value in dollars, by American industry, was 74 billion — only 23 billion of which was considered in the survey data reported and from which the total water consumption of 150 billion gals. daily was estimated. In Canada in 1950, the added dollar value of manufactured articles is reported as 6.3 billion dollars. Using the ratio of added dollar value between Canada and United States figures we arrive at a Canadian total 1950 water consumption of 10.6 b.g.d. (Imperial) and industrial consumption of about 1.8 b.g.d., which checks well with figures based on the population ratio. If MacKichan's estimate of 77 b.g.d. (U.S.) being used by U.S. industry is used, then Canadian industrial use would be about 5½ to 6 b.g.d. (Imperial).

However, available Canadian data are meagre and there are many discrepancies, particularly with regard to available information on water use by Canadian textile and pulp and paper industries. It is doubtful if we are justified in using American figures directly since water demands are so dependent upon basic industry of which there is probably a higher percentage in the United States. In the latter country there is also probably a larger percentage of water used for steam electric power generation than in Canada, where hydro-electric development is so great.

It is hoped in the future to obtain more detailed information on Canadian industrial use. H. R. Hooper studied 75 Canadian municipal water systems in 1948 and reported that 53 attributed increased consumption to industrial use. He found industrial consumption in one locality had increased over a 10-year period from 26 to 33 per cent, with total consumption increasing 20 per cent. Where 100 per cent metering was carried out, *per capita* consumption decreased, but industrial use was still 30 per cent. In survey studies to date we have found a number of municipalities where industrial use was much higher, some using 60 per cent or more of the total pumpage. Table II shows some of the information obtained from a questionnaire survey of the Canadian textile, pulp and paper, and leather industries carried out in 1947. It will be noted that while the leather industry (30 plants) used an average of about 111,000 gals. daily, consumption varied widely from 2,000 g.p.d. to 817,000 g.p.d. Process use by this industry averaged 80 to 89 per cent, cooling 10 per cent. The pulp and paper industry (67 plants) reported wide variation in consumption, with processing use varying from 35 to 97 per cent and cooling use from almost *nil* to 60 per cent. Roy Green, in his recent survey of some 3,000 plants or 1.3 per cent of total plants in the United States, reported on the percentage use for the five major industrial applications, namely, (1) cooling, (2) boiler feed, (3) processing, (4) sanitary, and (5) other uses. He reports that, for small industries, processing accounted for 36 per cent and cooling for 26 per cent, in the larger industries for 32 per cent and 54 per cent, respectively. The figures shown in Table II on the water use by the large Canadian pulp and paper industry differ widely from Green's figures on corresponding use in large United States industries. Considerable variation is to be expected in any questionnaire data, as it is often difficult even for the industry to make an accurate breakdown of the percentage water used for each application, since in many industries cooling water may be used for processing or *vice versa.*

The figures quoted are only rough estimates and are presented merely to indicate the enormous demands of industry, the variability of the demands due to type and size of the

industry, and the difficulty in obtaining accurate data. However, it is apparent that with such large demands, even if treatment costs are low, the overall cost of water as a raw material becomes appreciable in many industries.

Quality

It is not proposed to discuss in detail the various needs and demands of industry for quality in the water. Each industry has its own peculiar demands. Some have attempted to define the upper limits permissible in water quality for the production of certain materials. Table III shows some of the demands of the textile and paper industries. It will be noted some of these are quite stringent. Even though many industries do not have such narrow limits on quality, all users have common problems with corrosion, scaling, and slime growth, all of which are dependent upon water quality.

The following are the major constituents and properties of a natural water which determine its quality and affect its industrial use:

pH, acidity, and alkalinity are important in many process reactions. They are a measure of various equilibria in water and largely determine its corrosiveness or its tendency to cause scaling. In many processes, *pH* must be controlled within narrow limits.

Colour, which is often due to vegetation but may be caused by dissolved metals such as iron, is detrimental to many processes. It is often present as a colloidal, difficult-to-remove suspension, which stains white products such as fine paper, textiles, and pharmaceuticals. It may also cause undesirable side reactions with some organic materials.

Temperature of the water is very important for cooling purposes and for this reason many industries prefer the cooler ground waters to surface waters.

The *alkaline earth salts* such as carbonates, bicarbonates, sulphates, and chlorides of calcium and magnesium are without doubt the most recognized trouble makers. The solution of natural gypsum, dolomite, and limestone produces the well-known hardness in water which destroys soap and causes scaling in heating units, condensers, boilers, etc. The deposition of these salts in processing equipment and on products such as textiles, leather, and paper seriously deteriorate final product quality. Probably most treatment of water today is to remove or reduce the amount of these salts.

Table III.—Tentative Specifications

	Water Quality for Rayon Fibre Manufacture and Rayon Material Processing*	Water Quality for Fine Paper Manufacture, L,T.A.P.P.I. Tentativ. Standard E600 S44
	Permissible Upper Limits, p.p.m.	
Turbidity	1.0	10
Colour	5	5
Free carbon dioxide	—	10
Total hardness as $CaCO_3$	10	100
Calcium hardness as $CaCO_3$	—	50
Dissolved solids	200	200
Alkalinity as $CaCO_3$	75	75
Iron (Fe)	0.05	0.10
Manganese (Mn)	0.02	0.05
Aluminum (Al)	0.25	—
Heavy metals	0.01	—
Silica (SiO_2)	10	20
Residual chlorine as Cl_2	—	2.0
Non-living organic matter	5	—
Microbiological growth	0 (sterile)	—

*Miller, L. D., Rayon Textile, No. 26, p. 95 (Mar., 1947); No. 28, p. 130 (April, 1947).

Alkali salts such as sodium and potassium bicarbonates, sulphates, and chlorides are very soluble in water and are found in large quantities in the waters from certain areas such as the prairies and in some ground waters. If in large amount they can give rise to problems in boilers, in personal use, and in irrigation. In aqueous solution they are quite corrosive and difficult to remove.

The *metal content* in waters is generally quite low. However, some metals, such as iron and manganese, may be very detrimental in certain industrial uses, even when only 'traces' are present. In larger amounts they will cause staining by precipitation of oxides or they may react chemically with the product to lower its quality. Traces of other metals, such as copper and zinc, are often found in waters and may play a more important role in many problems than is at present recognized. As shown in Table III, the desirable limits on iron, manganese, aluminum, and heavy metals for fine textile production are quite low.

Sulphates and chlorides, if present in excessive amount, increase the corrosiveness of a water and, since they are often associated with the alkaline earths, cause permanent hardness which produces a hard, difficult-to-remove scale in boilers and other heating equipment.

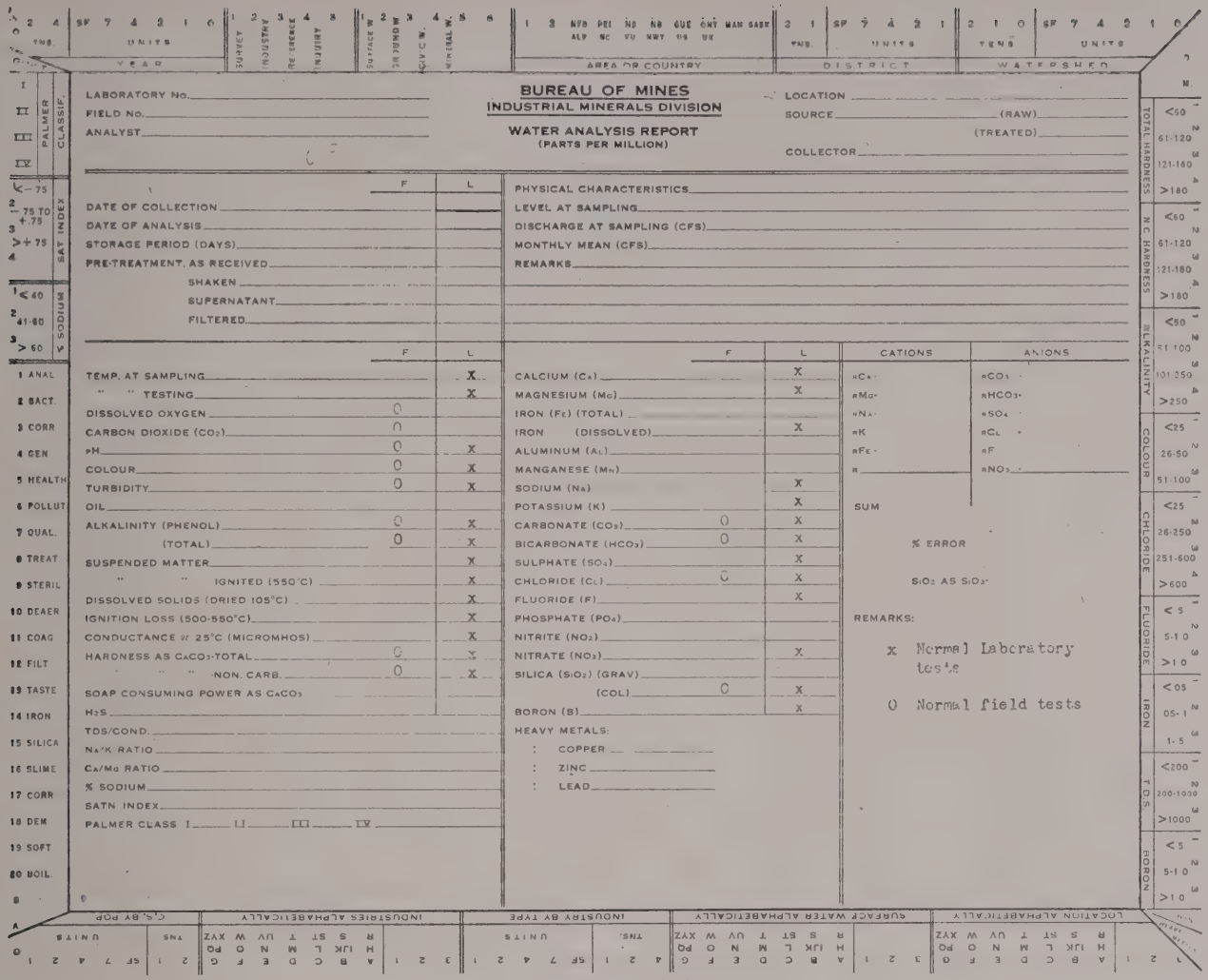
Nitrates and fluorides appear to affect industrial use very little. They have been found, however, to be important as regards health. High nitrate content in a water may indicate pollution and may cause a disease which has been fatal to babies. Fluorides are reported as having a beneficial effect in the development of teeth in the young.

In late years, *silica* in water has been found to be very important to certain industries. In high-pressure boilers, the silica is considered to distil over with the steam and may deposit on turbine blades, resulting in costly breakdowns and repairs. Silica can cause pin-holes in paper and affect other products in various ways. Its actual form in water is not altogether clear but it appears to be present either as colloidal silica or as a dissolved silicate. Silica depositing with the alkaline earth salts produces a very hard scale.

Trade wastes may add phenols, cyanides, chromates, and many other inorganic and organic compounds to a water, all of which may give rise, even when present only in traces, to serious problems in the growth of bacteria, algae, and slimes and increase other treatment problems and costs.

The importance of quality is demonstrated by the number and complexity of the water treatment facilities in various industries. Green, in his survey of United States industry, reports that 80 per cent of the larger plants treat the water and that 30 per cent of the small plants find treatment necessary. It is reported that one American water treatment firm sold, in five years, about 50 million dollars worth of equipment.

In the mining industry, water is used largely for power, as in steam boilers and for transportation. The importance of water quality in boiler operation is well known. For many years, water has been the main agent for separating precious metals



BUREAU OF MINES
INDUSTRIAL MINERALS DIVISION
WATER ANALYSIS REPORT
(PARTS PER MILLION)

LABORATORY No.
FIELD No.
ANALYST
LOCATION
SOURCE (RAW)
(TREATED)
COLLECTOR

F	L		F	L	CATIONS	ANIONS
		DATE OF COLLECTION				
		DATE OF ANALYSIS				
		STORAGE PERIOD (DAYS)				
		PRE-TREATMENT, AS RECEIVED				
		SHAKEN				
		SUPERNATANT				
		FILTERED				
	X	TEMP. AT SAMPLING		X	Ca	CO_3
	X	" " TESTING		X	Mg	HCO_3
0		DISSOLVED OXYGEN			Na	SO_4
0		CARBON DIOXIDE (CO_2)		X	K	Cl
0	X	pH			Fe	F
0	X	COLOUR		X	R	NO_3
0	X	TURBIDITY		X	SODIUM (Na)	
		OIL		X	POTASSIUM (K)	
0	X	ALKALINITY (PHENOL)	0	X	CARBONATE (CO_3)	SUM
0	X	(TOTAL)	0	X	BICARBONATE (HCO_3)	% ERROR
	X	SUSPENDED MATTER		X	SULPHATE (SO_4)	
	X	" " IGNITED (550°C)		X	CHLORIDE (Cl)	SiO_2 AS SiO_3
	X	DISSOLVED SOLIDS (DRIED 105°C)		X	FLUORIDE (F)	
	X	IGNITION LOSS (500-550°C)			PHOSPHATE (PO_4)	
	X	CONDUCTANCE @ 25°C (MICROMHOS)			NITRITE (NO_2)	
0	X	HARDNESS AS $CaCO_3$-TOTAL		X	NITRATE (NO_3)	
0	X	" " NON. CARB.			SILICA (SiO_2) (GRAV)	
		SOAP CONSUMING POWER AS $CaCO_3$		X	(COL)	
		H_2S		X	BORON (B)	
		TDS/COND.			HEAVY METALS:	
		Na/K RATIO			: COPPER	
		Ca/Mg RATIO			: ZINC	
		% SODIUM			: LEAD	
		SATN. INDEX				
		PALMER CLASS I II III IV				

REMARKS:

x Normal Laboratory tests

0 Normal field tests

Figure 2.—Water analysis report.

WATER SERVICE RECORD

MUNICIPALITY
INDUSTRY
SOURCE(S)
SAMPLED
OFFICER
SURVEY DATE
REMARKS:

POPULATION SERVED: MUNICIPALITY PROPER
OUTSIDE SERVICES
INDUSTRIES SERVED: 1 2 3
4 5 6 7
8 9 10 11

CONSUMPTION: (GPD)

PUBLIC	INDUSTRIES	TOTAL
AV.		
MAX.		
MIN		

% INDUSTRIAL USE
PUMPING EQUIPMENT:
RESERVOIRS:
METERING:

SERVICE PROBLEMS
1. CORROSION
2. BOILERS
3. TASTE, ODOR, ETC.:
4. PROCESSING

PLANT AND TREATMENT

(A) FLOW SHEET
(B) TREATMENT CHEMICALS, AMOUNT, Etc.
REMARKS:

Figure 3.—Water service record.

and the more valuable ores from their gangue. Nowadays, the quantities used in the concentration of ores by flotation is very large. Waters used for flotation usually must be alkaline. If not naturally alkaline they are made so by the use of chemicals such as lime or soda ash. The water should also be low in organic matter and not have too high a dissolved solids content. Of course, corrosive or scale-forming waters will cause serious problems in many mining and metallurgical operations.

Present Studies

In 1934, the Mines Branch began a survey of Canadian water resources, particularly as regards the quality available for use by industry. This study included spot sampling and analysis of major surface and municipal water supplies. The results have been reported in Mines Branch Bulletins. Owing to the war, the work was discontinued in 1943. This study did not consider seasonal variation, and was by no means a complete coverage, nor were the analyses of major supplies always as complete as desirable.

Increasing demands for information led to the present survey, begun in 1946. One major watershed or drainage basin is studied as completely as facilities permit for a period of one year. In the area selected, thirty sampling locations are chosen and monthly samples are collected, with extra samples being taken whenever possible at periods of high and low water. Occasionally, when the importance of the supply warrants it, daily sampling with compositing thrice monthly is employed. During the summer months, the area is travelled with a mobile laboratory and extra water samples are collected. At this time, municipal waters within the watershed are sampled and information on the system obtained. Tests which are carried out in the field laboratory indicate if storage and shipment is seriously affecting the analytical results on the monthly samples. With some waters, storage causes major changes in quality. This possibility should be kept in mind when considering many published results.

Figure 2 shows the card at present used for recording analytical data in the Industrial Waters Section. The determinations marked with a cross are normal procedure giving an analysis which is quite complete and which permits determining the analytical error. It will be noted that, by means of punching, the cards can be filed and sorted as to type or location of water. For example, cards for rivers can be sorted out by name, location, or province, or a province can be broken down into 39 districts. Analytical data, acceptable as to accuracy, from other laboratories is also being recorded on these cards.

Figure 3 shows the reverse side of some of these cards and indicates the information at present being obtained on municipal systems. The card is designed so that it can be used also for recording data on individual industries. It will be possible to code 69 different industries by type or alphabetically.

The Ottawa River watershed, all major areas of British Columbia, the St. Lawrence River-Great Lakes basin from Montreal to Parry Sound, and the drainage basins of the North and South Saskatchewan rivers, have now been studied and some of the results are already in the hands of the printer.

It will be some time before all of Canada has been so studied. To date, about 5,000 analyses have been completed. In the heavily industrialized area of southern Ontario alone there are about 225 municipalities with organized water systems, some of them having as many as 12 different water supplies.

The importance of adequate information on the seasonal variation of surface supplies is little appreciated. Figure 4 shows graphically some of the data obtained on the Rideau river at Ottawa during the period December, 1946, to December, 1947. It will be noted that the curves for total dissolved solids and total hardness are almost parallel, decreasing markedly with increasing flow. Turbidity follows closely the flow record. The Rideau river is fed in its upper reaches by a fairly soft water from heavily wooded areas; in its lower portion, it flows through almost flat clay and loam country where little forestation remains. In the spring, the melting snow and the seasonal precipitation flow rapidly from these lands into the river. The water has little chance to dissolve material but it carries in suspension considerable clay, as is indicated by the rapid increase in turbidity; thus, in the spring, the river water is fairly low in hardness and dissolved solids. In periods of low flow, however, the river is to a large extent fed by ground water from a limestone and clay area and run-off is from very slow streams, the waters of which have had considerable contact with the clay land. As a consequence, there is a marked increase in total hardness and dissolved solids.

Figure 5 shows the Ottawa river at Ottawa for the same period. This is a large river with many tributaries rising in heavily wooded northern portions of the country. In this river, turbidity, hardness, and dissolved solids all increase with increased flow. Variation in the discharge is not so marked, that is, the run-off is not so rapid, as indicated by the gradual slope in the discharge curve, It will be noted that while the discharge remains high over a three-month period, turbidity, dissolved solids, and hardness decrease fairly rapidly after the initial rise. This can be explained by considering that the initial increase, due to local run-off from the more settled, and less wooded, clay land in the lower

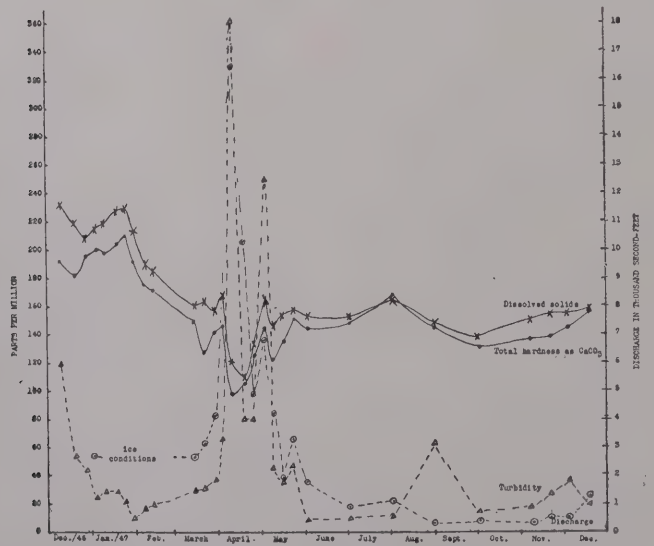


Figure 4.—Rideau river, near Ottawa.

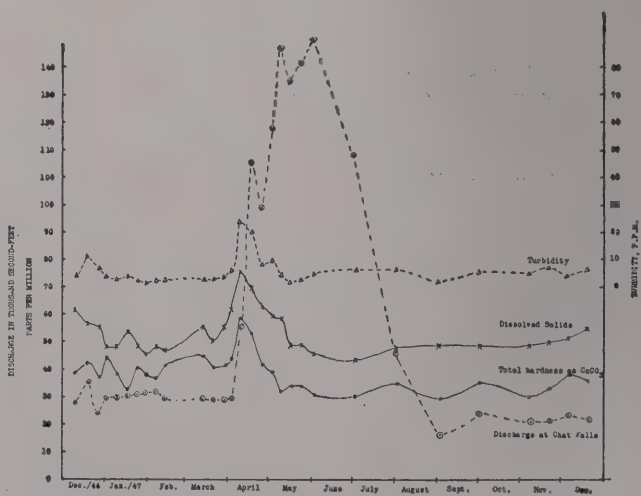


Figure 5.—Ottawa river, at Lemieux island.

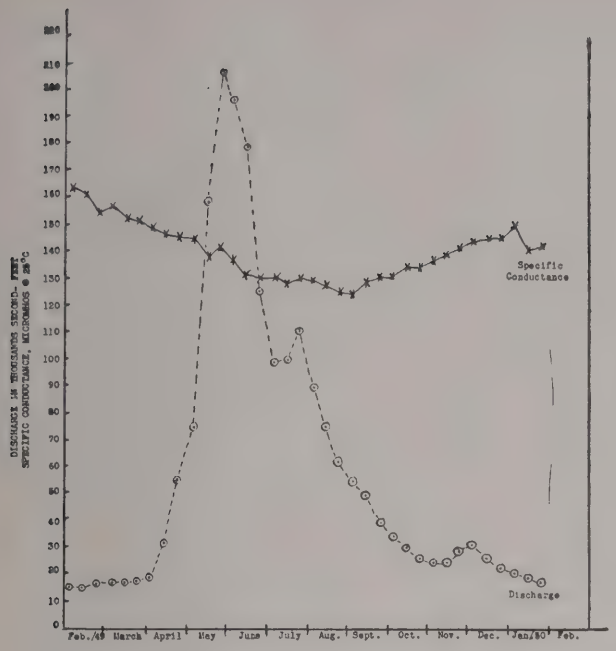


Figure 6.—Columbia river at Trail, B.C.

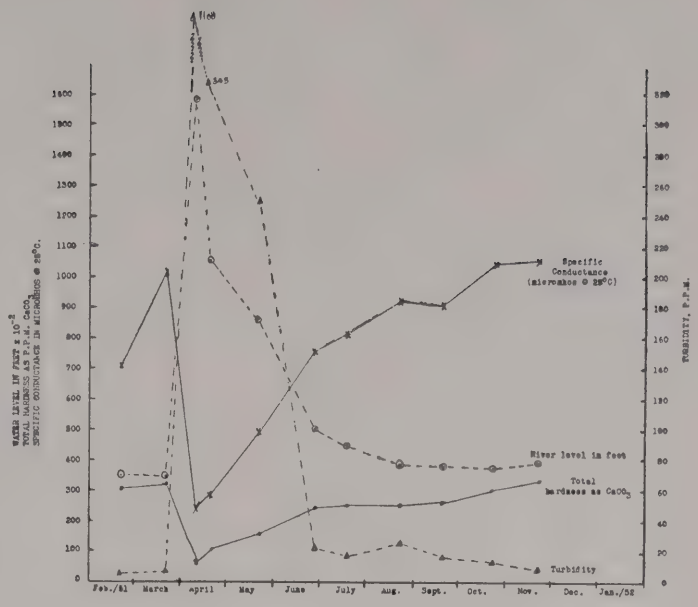

Specific Conductance (micromhos @ 25°C)
River level in feet
Total hardness as CaCO3
Turbidity
Feb./51 March April May June July Aug. Sept. Oct. Nov. Dec. Jan./52

Figure 7.—Battle river, near Unwin, Sask.

reaches produces a harder, dirtier water. As the snow melts in the upper wooded areas of northern Ontario and Quebec, the discharge remains high but the water is much softer and more highly coloured. At Ottawa, then, there are actually two floods, varying in quality.

Figure 6 shows the relationship between discharge and dissolved solids or specific conductance in the Columbia river at Trail, B.C. This also is a large river, with an extreme flood crest at times over 30 feet. However, the water quality remains markedly constant. From studies at other locations and on various tributaries in this watershed, and from the geology of the area, it is believed that this constancy is due to the different types of water entering the main river and to the settling out and mixing effect of the large Arrow lakes and Kootenay lakes prior to the water reaching the sampling location at Trail.

Figure 7 shows the type of seasonal variation occurring in the Battle river near Unwin, Saskatchewan. This river shows a rapid decrease in hardness and conductivity with quick rise in river level. Turbidity, however, rises rapidly. The interesting point here is the steady increase in conductance with decreasing flow, which is not due to increased hardness. It is probable here that, at periods of low flow, a larger percentage of the water entering the river is coming from local ground water and from sloughs and very small tributaries rising in the alkali prairie area. Thus the dissolved solids content increases, but not the hardness.

Similar curves can be prepared for all streams. It is not possible to predict actual quality changes until studies have been carried out on each watershed or river. While there is, in most rivers, a relationship between flow and water quality, studies so far have indicated that the relationship must be studied for a much longer period than one year before we can definitely say that the quality will be such and such when the flow is so many cubic feet per second. In this regard it is hoped, when the present survey studies are completed, to establish a few key stations on each of the major water sheds and to carry out analytical studies over a period of years.

Despite the studies going forward in many countries on water quality, water treatment, and related subjects, many problems still remain whose solution would be of value to the national economy and to industry. The lack at present of information on Canadian water requirements, usage, etc., is very apparent. There is still work to be done on treatment methods, on analytical procedures, on the relationship of materials in water to other factors, etc.

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DONALD J. MACNEIL, *President* S. C. MIFFLEN, *Secretary*

Presidential Address, M.S.N.S.

By DONALD J. MacNEIL

(*Annual Meeting, Mining Society of Nova Scotia, Sydney, N.S., July,* 1952)

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WHILE many parts of the world are embroiled in bitter internal and external disputes, Canada is experiencing a variety of aches that are more to be welcomed than rejected. We are having growing pains, and they hurt in spots. One of the spots, judging by the complaints we hear from time to time, is the Maritimes. A unit as large as Canada, with an area greater than the whole of Europe, cannot grow in all its parts with equal rapidity. The growth of south-central Canada during the past twenty years has been spectacular. It is obvious, however, that the industrial expansion of that area will, directly and indirectly, benefit other parts of the country. As the population grows, so too does the demand for more fuel, more electricity, more food, and more clothing. Naturally, every effort will be made by the people in that part of Canada to obtain those things as close to home as possible; but, with continued growth, the demand will greatly exceed the local supply, and thus, eventually, all parts of the country will feel the benefits of the tremendous developments that are under way in those local areas.

As far as the Maritimes are concerned, we who now live here should perhaps make certain changes in our conception of the future. I think we are in need of some real practical optimism; first, in regard to the things that we have, and second, in regard to the things that we can get. There is a feeling prevalent among our people that the acquisition of certain standards, the development of proposed industries, the discovery of new resources, are too good to be true. If some individuals have the temerity to propose *today* a programme that could materially expand our industrial wealth, there are too many among us who belittle and discourage such opportunities unless they can see results tomorrow.

All of us, I am sure, agree that it is more desirable to permit private enterprise to play the major part in the development of the Maritimes than it is to place that responsibility into the hands of the government. If, however, private enterprise, for reasons it considers sound, concentrates its efforts in some other section of the country, then we must resort to the best means at our disposal to excite an interest in these provinces, if in our opinion we have something to offer which will be of mutual benefit to all concerned. "Nothing ventured, nothing gained" is an old axiom that should guide us in some of our thinking along these lines.

Whether we like the results or not, we must admit that the world of this generation is new and startling, and that the circumstances under which we now live were molded by Science. We have suddenly found ourselves living in an ultra-scientific world. As far as our mineral resources are concerned, new technical methods have been devised in the fields of exploration, development, and operation. It is possible that in some respects they will not be as successful as the simpler procedures of yesterday; but we should bear in mind that those procedures were adequate and practical when people dealt with materials that were relatively easy to find and process. Presumably, all of the readily accessible resources that are presently in demand have been discovered, and in most cases developed. If more are needed, we must resign ourselves to the fact that they are not so readily accessible, and that it is going to require more effort, time, and money to discover and develop them. If we the people, through our government, decide that we should take the preliminary steps in some instances to interest private enterprise in the exploitation of our resources, we should be prepared to take advantage of all the modern scientific techniques these times demand.

May I insert here an example that comes to mind because it involves negotiations that are currently being conducted in this Province. As you may know, an attempt is under way to select a drilling site, with the hope that a commercial deposit of oil or gas will be discovered. If we turn to the United States for statistics, we will find that, in drilling a test outside of an already proven area, the chances of success are one in eighteen, if the location is selected without the best technological logic, whereas the chances are about one in seven (it works out one in 6.5) if the hole is located on the basis of geological and geophysical work. In the petroleum man's vocabulary, a 'wild-cat' is any hole drilled a mile and a half or more from a producing well. The proposed test in Nova Scotia will be about 140 miles from the Stony Creek, New Brunswick, oilfield. In getting this project under way there has been some criticism about the decision to do geophysical work before selecting the drilling site. Geophysical work is relatively inexpensive as compared with the $100,000 or so it would cost to drill the proper kind of test. With that much money at stake, it is imperative that the greatest care be taken in selecting the site to drill, notwithstanding the fact that a man once selected a successful oilwell location merely

by tossing his old battered hat over his shoulder! Some of the critics of the proposed attempts to find gas or oil in Nova Scotia belong to the category I referred to a moment ago; namely, the "too good to be true" school. Unfortunately for the more optimistic individuals, the critics are in a favoured position — they have six chances out of seven of being right, even if the test site is selected on the basis of geology and geophysics.

I would like to take this opportunity to congratulate a number of organizations that are doing excellent work in promoting the industrial growth of this Province. In doing so I may inadvertently neglect to give credit everywhere it is due; if so, I assure you that the oversight will be unintentional. I would first like to pay tribute to the Nova Scotia Department of Mines for the splendid assistance it is rendering, and has given in the past, on projects that require drills and portable mining equipment. This is but one phase of the Department's work. In addition, it is sponsoring a programme that is giving to the Province the very best geological advice available, due to arrangements it has made with Massachusetts Institute of Technology. The Centre which it established at Antigonish Harbour has, in the short period of its existence, received world-wide attention; some of the scientific conferences held there, and those that are planned for the future, are bound to reflect advantageously to the credit of Nova Scotia. The Department has also established a Mapping Division that is equal in efficiency to any we have in this country.

Another organization that merits praise and encouragement for the work it is doing is the Nova Scotia Research Foundation. Its establishment has already been justified, and out of it are bound to come results of great benefit to the Province. I would be remiss in my references to organizations that are of material assistance to the mineral industrial growth of Nova Scotia if I neglected to pay tribute to the Department of Trade and Industry, particularly in regard to the very earnest and persistent efforts it is making to establish an industry that will utilize the excellent deposits of high-calcium limestone, and of salt, that occur in eastern Nova Scotia.

The work that the Geological Survey of Canada, and other branches of the federal Department of Mines, is doing in this Province merits the highest commendation. The laboratory recently established in Halifax by the National Research Council will prove of great benefit to this part of Canada. Finally, the researches carried out by private mining companies in various parts of the Province, and the investigations conducted by the Nova Scotia Technical College, all will contribute, as they have in the past, to the growth and prosperity of our mineral industries.

The natural resources of any country mean nothing unless two other equally important and essential resources are taken into consideration. These two, indispensable if natural resources are to be of any value, are human resources and cultural resources. Among the former, of course, are included the abilities, the achievements, and the efforts of the people; the cultural resources are the tools, the roads, the eyeglasses, and all such things that can be utilized in the exploration and development of natural resources. Governments, in fact, are cultural resources; they can facilitate the development of natural resources, or, for generations dance on veins of by unwise policies, they can impede their utilization. Hottentots could gold; but, unless they recognized its worth and developed a method of extracting it profitably, it would represent something of no value to their native area as a natural resource.

We too, then, must judge our natural resources from the right viewpoint. We must have faith in the abilities and efforts of those who are attempting to evaluate them, and we should see that the right tools and the wisest policies are employed in their utilization. I, a few minutes ago, paid tribute to the organizations that are playing an active part in matters pertaining to the natural resources of Nova Scotia. Those organizations are, in fact, our cultural resources. With the exception of private mining companies, all of the groups to which I referred belong to us. We pay the bills, and we therefore have a right to be proud or critical of their efforts.

If a hundred needles are hidden in a haystack it may be relatively easy to find the first dozen; but the task becomes increasingly difficult as more and more are found. It is true that some may be discovered by accident, perhaps by someone who is sitting down on the job; but it is not the orthodox way to search for a needle, any more than it is the right way to look for mineral resources. It may be, in this relatively small Province, that the bulk of the 'needles' have been found; but I am of the opinion that more lie hidden. They may be hard to find, and we may have to uncover a few of them ourselves before we can convince private enterprise that it is well worth while to search for them.

MINING SOCIETY OF NOVA SCOTIA
DONALD J. MACNEIL, *President* S. C. MIFFLEN, *Secretary*

Coal in Canada, 1951

Industrial Requirements and Supply

By C. L. O'BRIAN *

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INTRODUCTION

NOTE.—All figures on the supply and consumption of coal in Canada cited in this paper were collected and reported by the Dominion Bureau of Statistics. The friendly and efficient co-operation of the officials of the Bureau is the foundation upon which it has been my privilege to rely in the preparation of this and two earlier papers (1). My thanks are due also to our own staff on the Dominion Coal Board, who spared neither time nor effort in helping to prepare this paper.

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STATISTICS are comparatively dull and, to many people, coal statistics are duller than most. Coal is, however, the one basic mineral upon which our whole way of life depends and it may be well if attention is drawn to this fact by an occasional statistical review of the coal position in Canada. The recent furore over the oil and gas discoveries has so obscured the basic position of coal in many minds that a little basic coal information may be useful. It is for this reason that, for the third successive year, a factual survey of the coal position is being presented.

In the previous papers (1) explanations have been given of the basic considerations in the preparation of these reports and some special phases of the overall picture have been examined. For these explanations, the earlier papers may be consulted.

The general picture of the supply and consumption of bituminous coal over the past twelve months follows fairly closely the pattern of the previous year, as shown in Table I, which sets out the overall story for each of the past five years. The data for the earlier years are corrected figures as issued by the Dominion Bureau of Statistics. Those given for 1951-52 are subject to later revision.

Comparing the figures for last year with those for 1950-51, it will be noted that there was a slight falling off in consumption and also in supply of both Canadian and United States coals. The supply of Canadian coal was lower by 200,000 tons, caused by a drop of production in the Maritimes of 144,000 tons and in the Western Provinces of 57,000 tons. The supply of United States coal was lower by some 1,500,000 tons, reflecting the reaction after the heavy build-up of storage piles last year following upon the stock depletion during the strike period in 1949-50.

It will be noted that the build-up during the summer months was heavier in the previous year but Table II shows that at December 1st our stocks of bituminous coal were some 2,000,000 tons higher than at December 1st, 1950. By April 1st, this overage had been reduced to a difference of 1,200,000 tons only. This increase in stocks at the close of navigation largely represented the build-up of storage to

TABLE I.— SUPPLY AND CONSUMPTION

YEAR	SUPPLY			CONSUMPTION		
	Canadian	United States	Total	Canadian	United States	Total
1937 Cal. year:						
Net tons....	15,836,000	12,130,000	27,966,000	15,464,000	11,662,000	27,126,000
%.....	56.6	43.4	100.0	57.0	43.0	100.0
1947-48:						
Net tons....	16,266,000	25,034,000	41,300,000	16,026,000	25,732,000	41,758,000
%.....	39.3	60.7	100.0	38.4	61.6	100.0
1948-49:						
Net tons....	19,468,000	25,027,000	44,495,000	18,628,000	24,573,000	43,201,000
%.....	43.8	56.2	100.0	43.1	56.9	100.0
1949-50:						
Net tons....	19,365,000	15,582,000	34,947,000	19,620,000	21,122,000	40,742,000
%.....	55.4	44.6	100.0	48.2	51.8	100.0
1950-51:						
Net tons....	18,708,000	23,699,000	42,407,000	18,730,000	21,239,000	39,969,000
%.....	44.1	55.9	100.0	46.9	53.1	100.0
1951-52:						
Net tons....	18,516,000	22,209,000	40,717,000	18,670,000	21,252,000	39,922,000
%.....	45.5	54.5	100.0	46.8	53.2	100.0

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(1) *Current Statistics on Coal in Canada,* by C. L. O'BRIAN; C.I.M., Trans., Vol. 53, 1950, pp. 397-406.
Coal in Canada: Particularly the Domestic Market, by C. L. O'BRIAN; C.I.M., Trans. Vol. 54, 1951, pp. 346-353.

protect the operation of the new, large coal-consuming plants coming into operation in Ontario. The power plant of the Ontario Hydro Electric Commission and the coke-oven battery of the Dominion Foundries and Steel Company are examples.

Imports of bituminous coal as reported in Table III show much the same general pattern and again demonstrate the importance of the water movement of bituminous coal over the Great Lakes.

The consumption of coal by railway locomotives (Table IV) continues to decline slightly, although it is still a substantial tonnage. The falling off is due to the increasing competition from both the Diesel and the oil burner, but it would also seem that increasing traffic is still maintaining the fleet of coal-burning locomotives in operation with only a small loss due to scrapping of the older engines.

The falling off for the year is some 136,000 tons over all of Canada but was not uniform over the different areas. Consumption in the Maritime Provinces actually increased by 10,000 tons, to 1,064,000 tons, but in Ontario and Quebec, with a consumption of 6,312,000 tons, there was a decrease of 339,000 tons. In the Western Provinces, on the other hand, locomotives used 3,411,000 tons, an increase of some 194,000 tons.

The sales of bituminous coal for household use are set out in Table V and show a substantial decline. Retail sales of anthracite and coke for household use also fell, from 4,225,000 tons in 1950-51 to 3,717,000 tons in 1951-52. The total decline in sales of both types of fuel for this market was 854,000 tons. Here, again, the situation was not uniform across the country. In the four Maritime Provinces, the total sales of household coal of 1,276,000 tons were 52,000 tons higher than in the previous year; in Central Canada there was a drop of 792,000 tons, for which the Toronto and Windsor areas were chiefly responsible; in Western Canada, the falling off was 300,000 tons.

It is difficult to find a clear explanation for this large drop in household coal consumption. At first thought, it might be considered as due to oil competition. Approximately 110,000 new oil burners were put in service across Canada in 1951. In that year, according to the Dominion Bureau of Statistics*, Some 81,300 homes were built. Supposing oil burners were installed in all these new homes, it would seem that some 28,700 of these burners would have been installed in older homes, thus replacing coal units. At an average use of 5 tons of coal per unit, this would only reduce coal consumption by 143,500 tons. a drop very much less than that actually experienced.

In Central Canada, it is likely that there was a substantial tonnage in consumers' basements last spring after the winter's higher than normal temperature. This spring, however, with the higher cost of fuel, it is probable that most cellars were empty. There may, therefore, have been a reduction in this hidden inventory, with a consequent drop in sales.

In Western Canada, the spreading use of propane is cutting into the sales of nut and pea coal for cooking and stove purposes.

Although there has been a drop in the consumption of coal for household heating, and a small decline in

**New Residential Construction*, 1951; published by the Dominion Bureau of Statistics.

TABLE II.— STOCKS OF BITUMINOUS COAL
(Net tons)

YEAR	MONTH	CANADIAN	UNITED STATES	TOTAL
1947	April 1	1,499,000	5,220,000	6,719,000
	December 1	2,567,000	10,225,000	12,792,000
1948	April 1	2,158,000	5,410,000	7,568,000
	December 1	2,803,000	13,205,000	16,008,000
1949	April 1	2,746,000	6,883,000	9,629,000
	December 1	2,981,000	7,720,000	10,701,000
1950	April 1	2,222,000	2,609,000	4,831,000
	December 1	2,561,000	10,015,000	12,576,000
1951	April 1	2,197,000	5,768,000	7,965,000
	December 1	2,478,000	12,114,000	14,592,000
1952	April 1	2,096,000	6,912,000	9,008,000

TABLE III.— IMPORTS OF BITUMINOUS COAL
(Net tons)

YEAR	BY RAIL	BY WATER	TOTAL
1945-46	8,446,770	14,472,544	22,919,314
1946-47	7,618,177	13,831,395	21,449,572
1947-48	8,955,289	16,079,114	25,034,403
1948-49	7,102,831	17,923,788	25,026,619
1949-50	4,434,474	11,147,519	15,581,893
1950-51	8,086,300	15,613,159	23,699,459
1951-52	6,742,155	15,466,979	22,209,134

TABLE IV.— CONSUMPTION OF BITUMINOUS COAL BY RAILWAY LOCOMOTIVES

YEAR	CANADIAN		UNITED STATES		TOTAL
	Net tons	%	Net tons	%	
1937	4,469,000	59.4	3,049,000	40.6	7,518,000
1946-47	4,005,000	32.8	8,198,000	67.2	12,203,000
1947-48	3,577,000	27.4	9,482,000	72.6	13,059,000
1948-49	3,850,000	30.4	8,829,000	69.6	12,679,000
1949-50	4,872,000	42.1	6,692,000	57.9	11,564,000
1950-51	4,353,000	39.8	6,579,000	60.2	10,932,000
1951-52	4,256,000	39.4	6,540,000	60.6	10,796,000

locomotive use, the consumption by industry has shown an increase over the previous year. From Table VI, it is apparent that the consumption this year has been the highest recorded since 1937 and it is probably the highest ever reached in Canada. The industrial use of coal in Canada will be examined in more detail after the more general picture has been presented.

The consumption of coal by the coke and gas industry is reported in Table VII. This tabulation has not been as satisfactory as it might have been since it includes two distinct components. There is the consumption of coal by plants producing gas and coke for sale, plus the consumption of coal for coking purposes only by the ovens attached to the big steel plants. Since there were only three of these steel plants and since only one of these uses Canadian coal, it has not been possible in the past to give a more detailed breakdown. Recently, however, there has been placed in operation the new battery of coke ovens of Dominion Foundries and Steel, Limited, at Hamilton. It is, therefore, possible to state that, for the fiscal year 1951-52, the tonnage of bituminous coal converted to coke in the ovens of the four steel companies was 3,656,000 tons. Consequently, the tonnage used in the same period by merchant coke and gas plants was 1,732,000 tons. The new ovens and the rehabilitation of existing plants during the past twelve months should lead to a larger consumption of coal in the coming year, and present information would justify an estimate of 4,400,000 tons in 1952-53 for steel plant consumption alone.

In Table VIII, the disposition of the remaining coal is recorded. Exports have increased over the previous year by a fair tonnage while consumption by other industries, which includes bunkering, colliery use, disposal of middlings, etc., has remained fairly steady.

Finally, the recapitulation in Table IX of the consumption for the past two years by types of consumers illustrates and accentuates the general trend.

The Industrial Market for Coal in Canada

The industrial use of coal in Canada has been primarily for the production, transfer, and application of heat. It has differed, therefore, from that in both the United Kingdom and the United States, where a large proportion of the coal is used in

Table V.—Sales of Bituminous Coal for Household Consumption

Year	Canadian		United States		Total
	Net tons	%	Net tons	%	
1937	4,654,000	67.6	2,229,000	32.4	6,883,000
1946-47	5,728,000	62.6	3,428,000	37.4	9,156,000
1947-48	5,645,000	60.3	3,721,000	39.7	9,366,000
1948-49	6,108,000	64.3	3,387,000	35.7	9,495,000
1949-50	6,302,000	66.7	3,147,000	33.3	9,449,000
1950-51	6,161,000	66.9	3,054,000	33.1	9,215,000
1951-52	5,838,000	65.8	3,031,000	34.2	8,869,000

Table VI.—Consumption of Bituminous Coal by Industry

Year	Canadian		United States		Total
	Net tons	%	Net tons	%	
1937	4,519,000	56.5	3,477,000	43.5	7,996,000
1946-47	3,282,000	33.5	6,528,000	66.5	9,810,000
1947-48	3,416,000	29.4	8,245,000	70.6	11,661,000
1948-49	4,305,000	35.5	7,836,000	64.5	12,141,000
1949-50	4,739,000	39.9	7,138,000	60.1	11,877,000
1950-51	4,617,000	38.8	7,286,000	61.2	11,903,000
1951-52	4,801,000	39.4	7,393,000	60.6	12,194,000

Table VII.—Consumption of Bituminous Coal by the Coke and Gas Industry

Year	Canadian		United States		Total
	Net tons	%	Net tons	%	
1937	1,191,000	32.1	2,425,000	67.9	3,716,000
1946-47	1,261,000	27.9	3,253,000	72.1	4,514,000
1947-48	1,079,000	22.8	3,636,000	77.2	4,715,000
1948-49	1,299,000	23.8	4,150,000	76.2	5,449,000
1949-50	1,406,000	27.1	3,789,000	72.9	5,195,000
1950-51	1,437,000	26.3	4,026,000	73.7	5,463,000
1951-52	1,378,000	25.6	4,010,000	74.4	5,388,000

Table VIII.—Consumption of Bituminous Coal by Other Industries, and Exports of Canadian Coal
(Net tons)

Year	Export of Canadian	Consumed by Other Industries		
		Canadian	United States	Total
1946-47	852,000	1,640,000	677,000	2,317,000
1947-48	758,000	1,552,000	647,000	2,199,000
1948-49	1,226,000	1,841,000	371,000	2,212,000
1949-50	447,000	1,864,000	355,000	2,219,000
1950-51	295,000	1,867,000	294,000	2,161,000
1951-52	493,000	1,904,000	278,000	2,182,000

thermal power stations to produce electricity.

In Canada, power for the operation of machinery is generally electrical, with little direct steam drive. The electricity used is largely produced from water power although some areas do not conform to the general pattern. In the calendar year 1951, there were produced some 55,590,622 M.kw.hr. in Canadian central electric stations from water power while thermal central stations produced 1,829.897 M.kw.-hr., or only 3.2 per cent of the total output.

Those industries, therefore, in which process heat is extensively used will be the largest coal consumers, whereas industries whose output depends mostly on power may use little coal. The demand for both process heat and the auxiliary building heating varies with the season so that, as reported in an earlier paper, there is a swing of as much as a million tons in industrial consumption between the coal consumed in the January-March quarter and that in July-September.

The bituminous coal consumption by industries by geographical area is presented in Table X, and it will be noted that Ontario and Quebec in this last year consumed over 9,-000.000 tons, or 74.7 per cent of the total. The variation from area to area can be studied in some detail from this table and, if a comparison with an earlier year is wanted, the same data for the year 1948-49 has been published in a previous paper*. The areas reported are those defined by the Dominion Bureau of Statistics in their *Annual Coal Statistics* and other coal reports.

There is also a wide variation of consumption by the various types of industry. Table XI gives data for the past fiscal year. These figures may be compared with those for 1945-46, as published in an earlier report*. The types of industry chosen are those reported by the Bureau in their standard reports, until the recent change and increase in the number of types. The consumption reported as *Iron and Steel* does not include the coal passed through the coke ovens but does cover all other consumption of coal by the steel plants.

Before discussing the consumption by areas more closely, it should be pointed out that it will be necessary sometimes to combine figures. It is a strict rule that many types of figures cannot be made public if they are built up from those of three

Current Statistics on Coal in Canada; Dom. Bur. Statistics.

TABLE IX.— CONSUMPTION OF BITUMINOUS COAL BY TYPES OF INDUSTRY
(Net tons)

INDUSTRY	1950 51			1951-52		
	CANADIAN	UNITED STATES	TOTAL	CANADIAN	UNITED STATES	TOTAL
Railway	4,353,000	6,579,000	10,932,000	4,256,000	6,540,000	10,796,000
Industrial	4,617,000	7,286,000	11,903,000	4,801,000	7,393,000	12,194,000
Household	6,161,000	3,054,000	9,215,000	5,838,000	3,031,000	8,869,000
Coke & Gas	1,437,000	4,026,000	5,463,000	1,378,000	4,010,000	5,388,000
Other	1,867,000	294,000	2,161,000	1,904,000	278,000	2,182,000
Exports	295,000	—	295,000	493,000	—	493,000
TOTAL	18,730,000	21,239,000	39,969,000	18,670,000	21,252,000	39,922,000

TABLE X.— INDUSTRIAL CONSUMPTION OF BITUMINOUS COAL BY AREAS, 1951-1952

AREA	CANADIAN		UNITED STATES		TOTAL NET TONS	PERCENTAGE OF TOTAL COAL
	Net tons	Percentage of total Canadian	Net tons	Percentage of total U.S.		
Nova Scotia	581,439	12.11	8,871	0.12	590,310	4.84
New Brunswick	792,216	16.50	10,350	0.14	802,566	6.58
Quebec	529,104	11.02	534,473	7.23	1,063,577	8.72
Montreal	823,424	17.15	725,197	9.81	1,548,621	12.70
Ottawa	9,603	0.20	974,322	13.18	983,925	8.07
Kingston	—	—	251,343	3.40	251,343	2.06
Toronto	4,321	0.09	2,180,766	29.50	2,185,087	17.92
Windsor	—	—	1,326,202	17.94	1,326,202	10.88
Northern	187,731	3.91	1,041,593	14.09	1,229,324	10.08
Western	202,135	4.21	320,831	4.34	522,966	4.29
Sub-Total	1,756,318	36.58	7,354,727	99.49	9,111,045	74.72
Manitoba	427,316	8.90	18,481	0.25	445,797	3.66
Saskatchewan	482,051	10.40			482,051	3.95
Alberta	298,161	6.21			298,161	2.45
Br. Columbia	463,807	9.66			463,807	3.80
TOTAL	4,801,308	100.00	7,392,429	100.00	12,193,737	100.00

TABLE XI.— CONSUMPTION OF BITUMINOUS COAL BY TYPE OF INDUSTRY, 1951-52

INDUSTRY	CANADIAN		UNITED STATES		TOTAL	
	Net tons	Percentage of total Canadian	Net tons	Percentage of total U.S.	Net tons	Percentage of total coal
Animal	108,030	2.25	198,117	2.68	306,147	2.51
Chemical	161,804	3.37	503,424	6.81	665,228	5.46
Iron and steel	240,065	5.00	879,699	11.90	1,119,764	9.18
Laundries	24,487	0.51	34,005	0.46	58,492	0.48
Mining	735,080	15.31	762,159	10.31	1,497,239	12.28
Non-ferrous	508,939	10.60	790,251	10.69	1,299,190	10.65
Non-metallic	108,029	2.25	93,145	1.26	201,174	1.65
Vegetable products	442,201	9.21	1,561,281	21.12	2,003,482	16.43
Wood	1,421,187	29.60	2,439,502	33.00	3,860,689	31.66
Electric power	1,051,486	21.90	130,846	1.77	1,182,332	9.70
TOTAL	4,801,308	100.00	7,392,429	100.00	12,193,737	100.00

or fewer individual firms. In areas, therefore, where the number of firms is small, it may be necessary to combine either the types of industry or the areas.

The classification by industry is somewhat empirical but was adopted by the Bureau and has only recently been changed for some other types of reports. In order to clarify the later data, the chief constituents of each of the main headings are listed in Table XII.

Maritime Provinces

There are some similarities in the industrial market for coal in the individual Maritime Provinces but there are also some wide differences.

Taking the four provinces as a whole, Table XIII gives the consumption by type of industry for the whole area. The *Wood* group is the largest user of coal and this is almost entirely by the pulp and paper industry, with five mills in the Province of New Brunswick and one in Nova Scotia. There had been a substantial consumption of coal in the pulp and paper industry in Newfoundland, but this has been converted to oil. The production of *Electric Power* was the second largest consumer, with Nova Scotia accounting for some 58 per cent and New Brunswick the balance. This thermal production of electricity was responsible for about 37 per cent of the total output of electricity in central stations, the balance being hydraulic. The *Iron and Steel* group consumption does not include the coal used in producing blast furnace coke nor — in view of the fact that there is only one company concerned — can this be given here. The greater proportion by far of this coal used by the iron and steel industry is, of course, used in Nova Scotia. *Mining*, with a usage of 41,000 tons, is fourth, although the coal used in coal mining itself is not included in this category. Some 24,000 tons were used in Nova Scotia, with the remaining 17,000 tons in New Brunswick.

There was a time when coal consumption was a matter of hand shovelling lump or mine-run coal into the boiler. This has long since passed away in the larger plants, although hand firing of lump coal is still practised in smaller boilers.

It may be of interest to note the size of coal consumed by industry. In the Maritime Provinces in this period, 1,248,000 tons or 89.5 per cent was either slack, stoker slack, or stoker, while the remaining 145,000 tons, or 10½ per cent, was sized coal.

TABLE XII.— CLASSIFICATION OF COAL USES, BY INDUSTRIES

ANIMAL	CHEMICAL	IRON AND STEEL
Meat packers	Chemicals	Foundries
Tanneries	Fertilizers	Steel fabricators
Fur dressers	Soaps	Primary iron smelters
Glue factories		
Fish canneries		

MINING	NON-FERROUS	NON-METALLIC
Cement plants	Copper	Oil wells
Mines other than coal mines	Lead	Oil refineries
	Zinc	Sand and gravel
	Nickel	Abrasives

VEGETABLE		WOOD	
Brewers & distillers	Greenhouses	Pulp & paper	Plywoods
Fruit & vegetable canneries	Textiles	Wallboard	Printing and publishing
Grain processors & elevators	Rubber goods	Roofing	
Food processors other than fish & meat	Tobacco	Barrels & kegs	Box factory
	Clothing	Wood treating	Sawmills
		Furniture	Fancy papers

POWER
Thermal (Steam)

TABLE XIII.— MARITIME PROVINCES
CONSUMPTION OF BITUMINOUS COAL BY TYPES OF INDUSTRY

INDUSTRY	CANADIAN	UNITED STATES	TOTAL	
	Net tons	Net tons	Net tons	Percentage of total coal
Animal	17,162	—	17,162	1.2
Chemical	15,681	—	15,681	1.1
Iron and steel products	207,339	—	207,339	14.9
Laundries	3,732	—	3,732	0.3
Mining	44,440	—	44,440	3.2
Non-ferrous metals	3,360	—	3,360	0.3
Non-metallics	971	—	971	0.1
Power	473,253	—	473,253	33.9
Vegetable products	95,013	—	95,013	6.8
Wood	512,704	19,221	531,925	38.2
TOTAL	1,373,655	19,221	1,392,876	100.0

TABLE XIV.— OIL CONSUMPTION BY INDUSTRY, ATLANTIC PROVINCES, 1950

INDUSTRY	TYPE OF OIL	IMPERIAL GALLONS		EQUIVALENT COAL AT 145 GALLONS PER TON	
BY ELECTRIC PLANTS	Heavy & medium oil	2,402,540		16,700	
	Diesel oil	4,250,000		29,300	
	TOTAL		6,652,540		46,000
BY INDUSTRY AS FUEL OR MATERIAL	Heavy & medium oil	39,044,088		270,000	
	Furnace oil	4,748,139		32,700	
	Diesel oil	4,989,301		34,400	
	Stove oil, etc.	8,547		60	
	TOTAL		48,866,975		337,160
	GRAND TOTAL		55,519,515		383,160

No recent figures on oil consumption are available, the last to hand being for the calendar year 1950. In that year, in the Atlantic Provinces, deliveries for consumption as reported by the refineries were as shown in Table XIV.

The industrial consumption in this area of both fuels expressed as tons of coal, and combining the figures for the two years, would be 1,754,326 tons and, of this, coal supplies some 78.2 per cent.

The new developments in industry in these provinces, such as the new cement plants, should raise the demand considerably over the next few years.

Ontario and Quebec

Because of the higher consumption in these two provinces and the differences in supply and types of consumption between different sections, the two provincial areas have been broken down into eight separate areas. The boundary lines do not follow the interprovincial boundaries and the individual areas will either be treated separately or grouped where this simplifies the data.

While there are differences between the type of use in the two areas of Quebec and Montreal, the similarities are close enough to justify the combining of the two for the purpose of this study. In Table XV, it will be seen that the industrial consumption of Canadian coal in these areas is almost as high as in the four Maritime Provinces. The *Wood* group, which is almost all pulp and paper, is by far the largest consumer of both Canadian and United States coal. This is the particular market that is so susceptible to secondary electric power and most of the mills in this area today maintain electric boilers on a standby basis. The *Vegetable Products* group is a large consumer and covers a wide selection of industries. *Mining,* consuming 380,090 tons, includes the large cement plant at Montreal as well as the mining areas of northern Quebec. It will be noted that the amount of coal consumed in *Power* production is very small. *Chemicals,* as would be expected, consume over 100,000 tons, while the *Non-Ferrous* group, comprising many works from the brass foundries of Montreal to the aluminum centre at Arvida, take over 200,000 tons. It will be noted that the large consumers of coal are those where heat is a vital element of the process. The demand for power is supplied from other sources.

Moving farther west, Kingston

TABLE XV.—QUEBEC AND MONTREAL AREAS
CONSUMPTION OF BITUMINOUS COAL BY TYPES OF INDUSTRY

INDUSTRY	CANADIAN		UNITED STATES		TOTAL	
	Net tons	%	Net tons	%	Net tons	%
Animal	39,205	2.9	4,859	0.4	44,064	1.7
Chemical	88,177	6.5	21,154	1.7	109,331	4.2
Iron and steel	16,891	1.3	146,122	11.6	163,013	6.2
Laundries	4,954	0.4	10,254	0.8	15,203	0.6
Mining	286,132	21.2	93,638	7.4	380,090	14.6
Non-ferrous metals	197,934	14.6	17,990	1.4	215,924	8.3
Non-metallics	346	—	10,653	0.8	10,999	0.4
Power	—	—	15,803	1.2	15,803	0.6
Vegetable products	179,128	13.2	208,117	16.6	378,245	14.8
Wood	539,461	39.9	713,060	58.1	1,270,521	48.6
TOTAL	1,352,528	100.0	1,259,670	100.0	2,612,198	100.0

TABLE XVI.—KINGSTON AND OTTAWA AREAS
CONSUMPTION OF COAL BY TYPES OF INDUSTRY

INDUSTRY	CANADIAN	UNITED STATES	TOTAL	
	Net tons	Net tons	Net tons	%
Animal	—	48,310	48,310	3.9
Chemical	5,197	45,152	50,349	4.1
Iron and steel	—	17,999	17,999	1.4
Laundries	—	2,209	2,209	0.2
Mining	—	188,582	188,582	15.3
Non-ferrous metals	—	25,530	25,530	2.0
Non-metallics	—	2,149	2,149	0.2
Power	—	—	—	—
Vegetable products	—	209,924	209,924	17.0
Wood	4,406	685,810	690,216	55.9
TOTAL	9,603	1,225,665	1,235,268	100.0

TABLE XVII.—TORONTO AND WINDSOR AREAS
CONSUMPTION OF COAL BY TYPES OF INDUSTRY

INDUSTRY	TORONTO		WINDSOR	
	U.S. coal Net tons	%	U.S. coal Net tons	%
Animal	115,406	5.3	28,195	2.1
Chemical	219,974	10.1	217,285	16.4
Iron and steel	431,250*	19.7	229,698	17.3
Laundries	18,013	0.8	1,950	0.2
Mining	246,950	11.3	169,528	12.8
Non-ferrous metals	127,357	5.8	2,560	0.2
Non-metallics	69,457	3.2	8,766	0.7
Power	101,362	4.6	13,527	1.0
Vegetable products	482,713	22.1	644,919	48.6
Wood	372,605	17.1	9,801	0.7
TOTAL	2,185,087	100.0	1,326,202	100.0

*Includes small tonnage of Canadian coal.

and Ottawa areas are combined in Table XVI, since, in this case also, there are marked similarities. *Pulp and Paper* is again the largest consumer, with 690,000 tons. Even in the Kingston area, this group consumed 21.1 per cent of the total industrial supply. The large tonnage shown for *Mining* includes coal for the cement plant in the Kingston district as well as for the developing of mines in that area. The consumption in the Ottawa area is, of course, very largely by pulp and paper plants, including wallboard, etc. It will be noted that there was a small consumption of Canadian coal in this past year, representing largely token shipments to markets supplied in past years that may, we hope, once again be supplied with coal from our eastern mines.

The Toronto and Windsor areas are the largest centres of industry of all Canada and might be expected to be much the largest consumers of coal. It is true that the Toronto area, requiring over 2,000,000 tons, is, in fact, the largest user in Canada, but the difference from other areas in coal consumption is much less than one would expect from the index of industrial output. The reason for this is that a very large portion of the industry in this area is power consuming and not particularly dependent upon heat. Hydro power runs the machines and coal or other fuel is used for heating buildings, water, and other similar tasks. Both these areas use American coal almost exclusively although there is a small tonnage of Canadian coal reported in the *Iron and Steel* group in Toronto. To prevent identification, it has been merged with the American coal.

The consumption in both Toronto and Windsor areas is given in Table XVII. There is a wider diversification among the different industries in the Toronto area than has been evident in any of the other areas. The *Vegetable Products* group, including brewers and distillers, canneries, greenhouses, etc., are large coal users, with *Iron and Steel* a close second. Even in this highly industrialized section of Toronto, the *Pulp and Paper* group reports a large consumption. It is also interesting to note that such a large industrial concern as General Motors reports as a minor consumer of coal in the *Iron and Steel* group.

Moving farther west to Windsor, there is immediately less diversity and greater concentration in certain fields. Polymer, with its large coal consumption, contributes to the total for the *Vegetable Products* group, while Brunner-Mond, Canadian Industries Limited, Ford, Chrysler, Dominion Sugar, and other large industries fall naturally into their own

TABLE XVIII.— NORTHERN AND WESTERN AREAS
CONSUMPTION OF COAL BP TYPES OF INDUSTRY

INDUSTRY	CANADIAN		UNITED STATES		TOTAL	
	Net tons	%	Net tons	%	Net tons	%
Animal	—	—			—	—
Chemical	—	—	417	—	417	—
Iron and steel	—	—	56,948	4.2	56,948	3.2
Laundries	—	—	1,534	0.1	1,534	0.1
Mining	8,234	2.1	52,743	3.9	60,97	3.5
Non-ferrous metals	100,440	25.8	617,977	45.4	78,417	41.0
Non-metallics	—	—	1,441	0.1	1,441	0.1
Power	—	—	—	—	—	—
Vegetable products	531	0.1	12,265	0.9	12,796	0.7
Wood	280,661	72.0	619,099	45.4	899,760	51.4
TOTAL	389,866	100.0	1,362,424	100.0	1,752,290	100.0

TABLE XIX.— ONTARIO AND QUEBEC
TOTAL COAL CONSUMPTION BY TYPES OF INDUSTRY
AND COMPARISON OF PERCENTAGES BY AREAS

INDUSTRY	PERCENTAGE CONSUMED IN EACH AREA					TOTAL CONSUMPTION	
	QUEBEC AND MONTREAL	KINGSTON AND OTTAWA	TORONTO	WINDSOR	NORTHERN AND WESTERN	NET TONS	%
Animal	1.7	3.9	5.3	2.1	—	235,975	2.6
Chemical	4.2	4.1	10.1	16.4	—	597,329	6.6
Iron and steel	6.2	1.4	19.7	17.3	3.2	898,908	9.9
Laundries	0.6	0.2	0.8	0.2	0.1	38,914	0.4
Mining	14.6	15.3	11.3	12.8	3.5	1,046,127	11.5
Non-ferrous metals	8.3	2.0	5.8	0.2	41.0	1,089,788	11.9
Non-metallics	0.4	0.2	3.2	0.7	0.1	92,812	1.0
Power	0.6	—	4.6	1.0	—	130,692	1.4
Vegetable products	14.8	17.0	22.1	48.6	0.7	1,737,597	19.1
Wood	48.6	55.9	17.1	0.7	51.4	3,242,903	35.6
						9,111,045	100.0

groups. *Wood Products* play a very small part in this highly industrialized section of Ontario and the same is true of the other raw material groups of *Non-Ferrous Metals* and *Non-Metallics*.

Finally, the Northern and Western areas, or perhaps better the frontier areas, are combined in Table XVIII. Once again the pattern of the Quebec area is repeated, with a heavy consumption by the *Pulp and Paper* industry. The *Non-Ferrous Metals* group includes, of course, the large smelter of the International Nickel Company as well as other non-ferrous refineries. Canadian coal again forms a substantial portion of the total supply, with both eastern and western coal participating. Consumption by other users is comparatively small, with the *Iron and Steel* group the only one of any consequence.

To summarize this Central Area, Table XIX repeats the percentages of the previous tables and shows also the total tonnage consumed by each industry. The Table brings out clearly the variation from area to area and also indicates what types of industry are concentrated in each of the areas.

In dealing with the Maritime Provinces, data were given on the proportion of various coal sizes used by industry. In Ontario and Quebec there is a further break-down available, since the Bureau of Statistics reports the consumption of imported stoker coal and also splits the imported coal into high and low volatile. This break-down is shown in Table XX.

It will be noted that while only 10 per cent of the coal used by industry in the Maritimes was sized, in this Ontario-Quebec area almost 15 per cent was sized. This probably reflects the greater number of small coal-consuming industries and also those whose boilers are used chiefly for heating on a seasonal basis, in which cases the installation of stokers may not be economic.

It is of interest to compare oil consumption for 1950 in the Provinces of Ontario and Quebec with the total coal consumption of 9,111,045 tons (Table XIX). Including the small quantity of oil used in electric plants, the reported figures and their equivalent in tons of coal are as given in Table XXI.

Thus, the total consumption of coal and of oil expressed as tons of coal (2,282,700 tons) was 11,393,745 tons, of which coal supplied 80 per cent.

TABLE XX.— BREAK-DOWN OF ONTARIO AND QUEBEC COAL CONSUMPTION BY SIZES OF COAL

	NET TONS		PERCENT
Canadian slack and stoker coal	1,705,000		18.7
U.S. high volatile slack coal	4,820,000		52.9
U.S. low volatile slack coal	245,000		2.7
TOTAL, slack, etc.		6,770,000	74.3
U.S. high volatile stoker coal	946,000		10.4
U.S. low volatile stoker coal	54,000		0.6
TOTAL, U.S. stoker coals		1,000,000	11.0
Canadian sized coal	47,000		0.5
U.S. high volatile sized coal	1,156,000		12.7
U.S. low volatile sized coal	138,000		1.5
TOTAL, sized coal		1,341,000	14.7

TABLE XXI.— OIL CONSUMPTION BY INDUSTRY, ONTARIO AND QUEBEC, 1950

TYPE OF OIL	IMPERIAL GALLONS	EQUIVALENT COAL AT 145 GALLONS PER TON
Heavy and medium oil	276,832,426	1,989,200
Furnace oil	37,995,863	262,000
Diesel oil	11,423,119	78,800
Stove oil, kerosene, etc.	4,748,449	32,700
TOTAL	330,999,857	2,282,700

TABLE XXII.— MANITOBA AND SASKATCHEWAN CONSUMPTION OF COAL BY TYPES OF INDUSTRY

INDUSTRY	CANADIAN		UNITED STATES		TOTAL	
	Net tons	%	Net tons	%	Net tons	%
Animal	51,369	5.7	1,126	6.1	52,495	5.7
Chemical	1,536	0.2	—	—	1,536	0.2
Iron and steel	12,830	1.4	27	0.1	12,857	1.4
Laundries	15,222	1.7	—	—	15,222	1.6
Mining	144,795	15.3	13,385	72.4	158,180	17.1
Non-ferrous metals	102,201	11.3	—	—	102,201	11.0
Non-metallics	109,581	12.0	360	2.0	199,941	11.8
Power	351,906	38.7	523	2.8	352,429	38.0
Vegetable products	93,190	10.2	3,060	16.6	96,250	10.4
Wood	26,737	3.0	—	—	26,737	2.8
TOTAL	909 367	100.0	18,481	100.0	927,848	100.0

TABLE XXIII.— OIL CONSUMPTION BY INDUSTRY, MANITOBA AND SASKATCHEWAN, 1950

TYPE OF OIL	IMPERIAL GALLONS	NET TONS COAL AT 145 GALLONS PER TON
Heavy and medium oil	30,943,012	213,500
Furnace oil	1,337,756	9,200
Diesel fuel	20,345,004	140,300
Stove oil and kerosene	1,104,071	7,600
TOTAL	53,729,843	370,600

In the present and future years, the coal consumption of the new large thermal electric power stations at Toronto and Windsor will very materially change these figures. It is reported that they will consume over 800,000 tons of coal per year when in full operation.

The consumption of coal by industry in this part of Canada continues to increase and reports of surveys show that existing boiler plants are generally operating at or over nominal capacity. The coal requirements for this type of operation stress quality, uniformity, and sizing, and it has been considered wise economy to purchase premium coals as an insurance for trouble-free operation. The reported use of 1,000,000 tons of stoker coal is evidence of this necessity to secure the highest possible trouble-free performance from existing boiler installations. New plants, however, should soon move into the designing stage and the latest developments in combustion techniques, such as the cyclone burner, offer promise for a real widening in the type and quality of coal required.

Manitoba and Saskatchewan

These two Prairie Provinces are generally similar in type of industry and, as will be seen in Table XXII, their combined coal consumption is almost 1,000,000 tons. In these provinces, a large proportion of the coal used is lignite. In the year under review, consumption of bituminous coal amounted to only 220,000 tons, as compared with 707,000 tons of lignite or sub-bituminous coal.

Power production accounts for most of the coal being used in Saskatchewan. In the *Mining* group, there is, of course, the large cement plant at Fort Whyte, Manitoba, as well as the mining operations in northern Saskatchewan and Manitoba. The consumption by other industries is well diversified, although the *Chemical* and *Iron and Steel* groups use comparatively little coal.

The consumption of a large tonnage of lignite has been noted above. For industry, this coal is supplied in either a slack or stoker-slack form; it is not used to any extent as lump. This fact is reflected in the break-down by sizing. Of the coal used by industry in these provinces in 1950, 891,000 tons or 96 per cent was slack or stoker-slack while only 37,000 tons, or 4 per cent, was sized. This is a considerable change from the practice in the Maritimes and Central Provinces.

Oil consumption is, of course, fairly large in this area. In 1950, deliveries from the refineries to industry are reported as shown in Table XXIII.

The total, therefore, of coal consumed, and of oil deliveries as equivalent tons of coal, would be 1,298,448 tons, of which total coal supplied 71½ per cent. The cheapness and ready availability of the lignite coals are undoubtedly factors in retaining this market for coal.

Alberta and British Columbia

The area from the Alberta-Saskatchewan boundary to the Pacific coast actually has two differing areas of consumption, the Coastal market and the Interior. Unfortunately, these are not the geographical divisions nor are the statistics compiled on such a basis. It has appeared advisable, therefore, to report on the two provinces as one area and Table XXIV gives the information in detail.

It will be seen that consumption is concentrated in a few large types of industry and that *Mining* and *Power* together account for over 60 per cent of the total. The power usage is confined almost altogether to Alberta, where there are several thermal electric power stations. Mining is, of course, largely in British Columbia and this group includes also one cement plant.

This concentration in larger industries is also reflected in the break-down by sizes. Of the total (761,968 tons, see Table X), some 740,000 tons or 97 per cent was in the form of slack and stoker, while only 22,000 tons, or 3 per cent, was screened coal. This is, of course, a natural development when the western bituminous coal is taken into account. This is a friable coal and, in sizing, the proportion of fines is high. The largest outlet for the output from the mines is the railway market and this demands a sized coal. There is, consequently, an over-supply of fines and this tends to encourage the installation by industry of equipment designed for burning slack coal.

Coal consumption by the *Wood* industry is low when the number of large pulp and paper mills on the coast is considered. There are some nine of these mills and, at the time of the last survey, they were using fuel oil in an amount equivalent to some 375,000 tons of coal. The sawmills, of course, use sawdust, bark, and waste wood extensively, probably supplemented with some fuel oil.

TABLE XXIV.— ALBERTA AND BRITISH COLUMBIA
CONSUMPTION OF COAL BY TYPES OF INDUSTRY

INDUSTRY	CANADIAN	
	Net tons	%
Animal, iron and steel, and laundries	2,532	0.3
Chemical	49,549	6.4
Mining	250,733	33.0
Non-ferrous metals	103,452	13.6
Non-metallics	—	—
Power	217,417	28.5
Vegetable products	72,280	9.5
Wood	66,005	8.7
TOTAL	761,968	100.0

TABLE XXV.— OIL CONSUMPTION BY INDUSTRY, ALBERTA AND BRITISH COLUMBIA, 1950

TYPE OF OIL	IMPERIAL GALLONS	NET TONS OF COAL AT 145 GALLONS PER TON
Heavy and medium	78,248,996	539,600
Furnace	1,756,239	12,100
Diesel	15,369,484	105,900
Stove & kerosene	1,849,255	12,700
TOTAL	97,223,974	670,300

There are no exact data available on the wood waste burned in industry, but the deliveries of oil for industrial use in 1950, as reported, are shown in Table XXV.

Thus industry's consumption of coal and oil (assigning to the latter its coal equivalent) totalled 1,432,268 tons, of which coal itself supplied 53.2 per cent.

Natural gas is a growing factor and it is of interest to measure the impact of this comparatively new industrial fuel. Figures are not available on consumption by provinces but have been reported for some years on the Western Provinces as a whole. The reported supply of natural gas for industrial use in Western Canada has been:

1948	11,524,972 M. cu. ft.
1949	13,152,971 "
1950	16,346,948 "
1951	20,166,330 "

If these figures are combined with those already given for oil (for 1950) and for coal, the industrial fuel market for the four Western Provinces in terms of coal as tons or equivalent tons has been:

	Tons or Equiv. Tons	%
Coal	1,789,816	49.6
Oil	1,040,900	28.9
Gas	775,000	21.5
TOTAL	3,605,716	100.0

In Alberta, particularly, there is a large swing from coal to natural gas and this will probably continue. It is not possible to catalogue this in detail as it is a case of individual plants. It does not follow that coal will decline as the use of gas increases. New developments and new industries will change the overall situation, and it may well be that, in these changes, coal will continue to play a most important part.

CONCLUSION

In the foregoing pages, an attempt has been made to present a somewhat detailed picture of the industrial outlets for coal in Canada. It has necessarily been burdened with statistics for which an apology could be made were it not that they form the factual basis of the whole presentation. It is hoped that the report may be of some value to all who are interested in the Canadian fuel market.

The Effect of Water Corrosion and Shot Peening on Fatigue Strength of Mining Drill Rods*

By T. W. WLODEK†

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ABSTRACT

The results of research into the fatigue properties of *SAE 1080* carbon steel and nickel-chromium-molybdenum steel for mining drill rods are analysed and described in detail.

The effect of water corrosion on the fatigue strength of drill rods of these two steels in the as-rolled condition has been investigated and the beneficial influence of shot-peening on their dry and water-corrosion fatigue strengths determined by employing the method developed previously (1).

Eight *S-N* curves for as-rolled and shot-peened mining drill rods under both dry and water-corrosion conditions have been derived and their shape and relative position are discussed. The main features relating to field performance of the drill rods are analysed.

Purpose of Research and Outline of Programme

DRILLING holes in rock and ore by means of drill rods attached to a pneumatic drill is a basic procedure required in mining operations. Detonation of explosives placed in these holes is at present the only practical method of breaking down the rock and ore.

A drilling rod is supported by its bit end in the drilled rock. Its shank end, fixed in the drilling machine, transmits impact energy to the cutting edges of the bit. On the field performance of the drill rod depend a number of basic mining operations such as sinking, drifting, and stoping. The drill rod is therefore regarded as one of the most important tools used in the mining industry.

The cost of the drill steel, and the length of its working life during actual operations, have a direct bearing on the cost per ton of mined ore and rock.

The overall technical and economical performance of drill steel depends on its mechanical properties and metallurgical characteristics, that is, on its adaptability to forging, heat treating, and machining. These metallurgical characteristics have a direct bearing on the manufacturing procedure and field performance of the shank-end and the bit-end attachments of the drill rod.

In this paper we shall limit our interest to the drill rod as such. We shall study only the performance of that length of drill rod which lies between the bit attachment and the shank-end and is not usually exposed to the effect of the forging and heat-treating operations, which are carried out at both ends. Mechanical and metallurgical analyses of the attachment and shank end, and their field performance, will be given separately (2).

Simulated Service Tests

A specially developed method of fatigue testing mining drill steel was used for this research (1). In this method, the working conditions of mining drill rods are simulated as simply as possible; reversed bending is applied, under dry and water-corrosion conditions.

In drill steel, during actual working conditions, axial dynamic tension-compression stresses are superimposed upon the reversed bending and small torsion stresses. The exact magnitude of these stresses could be determined by stress measurements taken during drilling operations.

At present, the equal tension-compression loading under rotating bending would be the simplest choice for our simulated service tests. A similar procedure is applied in the fatigue testing of machine components (other than drill steel) and standard test samples; in such testing, for the sake of simplicity, rotating bending is usually chosen.

The effect of the superimposed static compression loads could be analysed separately on the so-called Goodman diagrams, but for drill steel the evaluation of fatigue characteristics under equal tension-compression should be quite satisfactory.

In our simulated service test and in actual field operations, the stress and number of cycles to failure — the so-called *S-N* relationship — are recommended for the evaluation of the dynamic characteristics of drill steel.

The obtained *S-N* relationship could be used for the development work of new drill steels, as well as for the study of the effect of changes in the manufacturing procedure. The effects of water corrosion, surface defects, and especially applied surface treatments such as shot peening, could also be analysed by that method.

Simulated service conditions could be repeated and duplicated at will, and therefore characteristic *S-N* relationships are reproducible and could be used either by the producer or the user of drill steel as a standard constant for the inspection of the quality of separate heats.

During drilling, the piston of the air drill strikes one end of the drill rod repeatedly, causing the other end of the bit, supplied with cutting edges, to advance into the rock.

All sections of the steel rod are not stressed simultaneously to the same magnitude; a wave of stresses having the velocity of sound in steel is produced by the impact of the piston and the back reaction of the rock. This resultant tension-compression axial wave of stresses is superimposed on a wave of bending stresses which is produced by the side vibration of the drill steel and the eccentric reaction of the rock on the drill bit.

The actual magnitude and type of the working stresses in drill steel can be determined by direct measurements employing *SR-4* electrical

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†Research Metallurgist, Division of Physical Metallurgy, Mines Branch, Ottawa.

(1) For references, see end of paper.

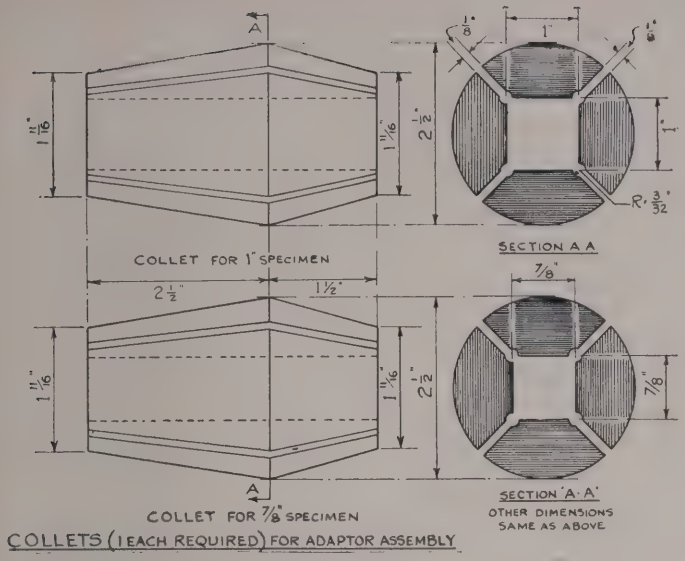
COLLET FOR 1" SPECIMEN
SECTION A-A
COLLET FOR 7/8" SPECIMEN
SECTION 'A-A'
OTHER DIMENSIONS SAME AS ABOVE
COLLETS (1 EACH REQUIRED) FOR ADAPTOR ASSEMBLY

Figure 1.

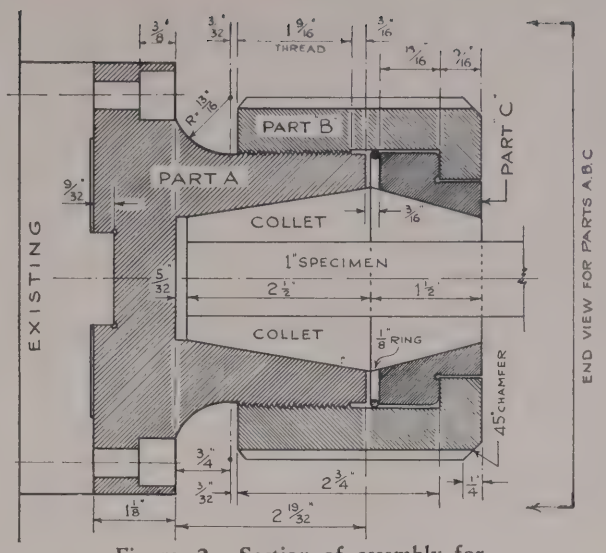

Figure. 2.—Section of assembly for adaptor.

strain gauges, or by using analytical methods if proper supporting conditions could be assumed (3, 4, 5).

Description of the Adaptors

Specially designed, self-centred adaptors were used for the work presented in this paper. The cross-section of the head of one of these adaptors, with four flat conical collets, is shown in Figures 1 and 2. The 4-in.-long flat collets, as compared to the previously used 2 3/32 in. collets, provide a much longer gripping surface.

Round nuts threaded in the opposite direction with conical self-centred collets provide the very desirable feature of self-tightening during rotation. This self-locking feature is important because any loosening of the bakelite and brass shims could be made up by further self-tightening, eliminating excessive vibration of the sample.

The design of adaptors shown in Figures 1 and 2 is a final one, and will be used during a subsequent series of investigations evaluating the dynamic characteristics of drill steels used at present by the mining industries.

Material Used

The mining and metallurgical industries are continually investigating the field performance of different types of drill steel. A large volume of data for different mining and operating conditions has been accumulated, together with information on block test performance. It would be very desirable from the research viewpoint, and useful in industrial development, to have reliable and reproducible technical data on the dynamic characteristics of various drill steels, such as endurance strength and fatigue limit under simulated service conditions. The present work is an attempt to obtain such data.

The test material used for the work described here consisted of 1-in. quarter-octagon drill steel made from the two most commonly used chemical compositions, namely, *SAE 1080* plain carbon steel and Ni-Cr-Mo steel, both in the as-rolled condition, with and without shot peening. The chemical composition of these steels is given in Table I and the mechanical properties in Table II. A complete mechanical-metal-

Table II.— Mechanical Properties of SAE 1080 and Ni-Cr-Mo Steels

Steel	Ultimate Tensile Stress p.s.i.	0.2% Proof Stress p.s.i.	0.1% Proof Stress p.s.i.	0.01% Proof Stress p.s.i.	Elongation % on 1 in.	Elongation % on 2 in.	Reduc. in Area, %	Charpy Impact (*55×10×10mm.; V-notch 2 mm. deep*) ft.-lb.	Brinell Hardness Number	Approx. Modulus of Elasticity p.s.i.
SAE 1080 Carbon..	130,970	70,600	67,800	59,200	16.3	12	23.2	11.6	244	29.6×10^6
Ni-Cr-Mo.....	167,930	110,500	101,200	72,870	13	9.5	30.9	6.0	302	28.0×10^6

Steel	Drill Rods, "as rolled"				Drill Rods, "shot-peened"			
	Fatigue Limit p.s.i.	Ratios: Fatigue Limit UTS	Ratios: Fatigue Limit 0.2%PS	Ratios: Fatigue Limit 0.01% PS	Fatigue Limit p.s.i.	Ratios: Fatigue Limit UTS	Ratios: Fatigue Limit 0.2% PS	Ratios: Fatigue Limit 0.01% PS
SAE 1080 Carbon.	35,050	0.27	0.50	0.59	44,700	0.34	0.64	0.75
Ni-Cr-Mo.....	47,800	0.29	0.44	0.67	64,750	0.39	0.59	0.89

Note: All results are the average of three tests, with the exception of fatigue limit determinations.

TABLE I.— CHEMICAL COMPOSITION OF DRILL STEELS

	SAE 1080 CARBON STEEL	Ni-Cr-Mo STEEL
Carbon.....	0.77 per cent	0.47 per cent
Manganese..	0.27 "	0.66 "
Silicon.....	0.19 "	0.26 "
Phosphorus..	0.017 "	0.026 "
Sulphur.....	0.025 "	0.020 "
Chromium...	0.10 "	0.37 "
Nickel.....	Trace	2.93 "
Molybdenum	Trace	0.21 "

lurgical examination of these two steels is given in a separate report (6).

Drill rods used in this reserch are taken from the regular production and therefore represent the average quality available on the maket.

The dynamic characteristics of the *SAE 1080* drill steel, in terms of stress and number of cycles to failure, are shown in Figure 3. In this chart, the curve *A* represents data for the as-rolled surface under dry conditions, and the curve *B* those for the as-rolled and water-corrosion condition. The curves *C* and *D* show the corresponding relations for shot-peened drill rods.

For Ni-Cr-Mo steel, the corresponding relations are shown in Figure 4 on curves *E*, *F*, *G*, and *H*.

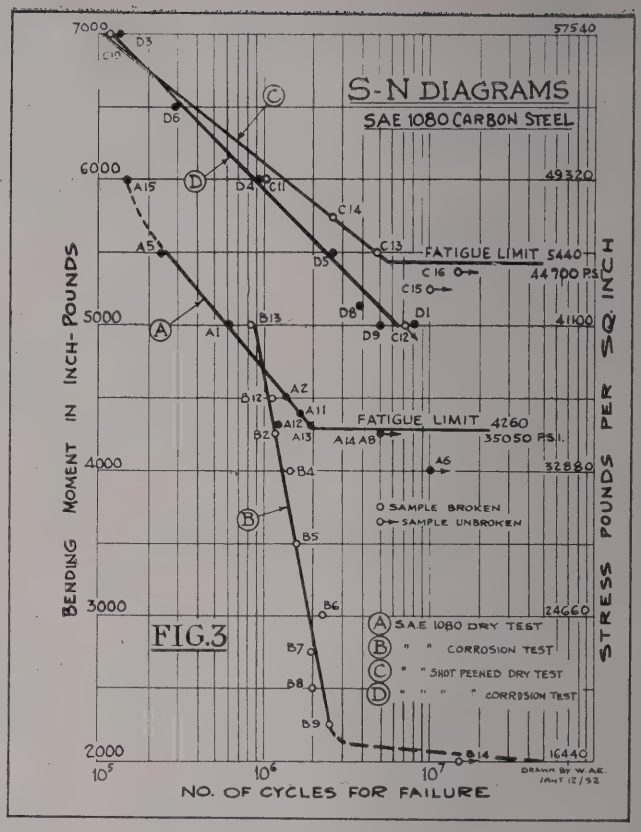

Figure 3.

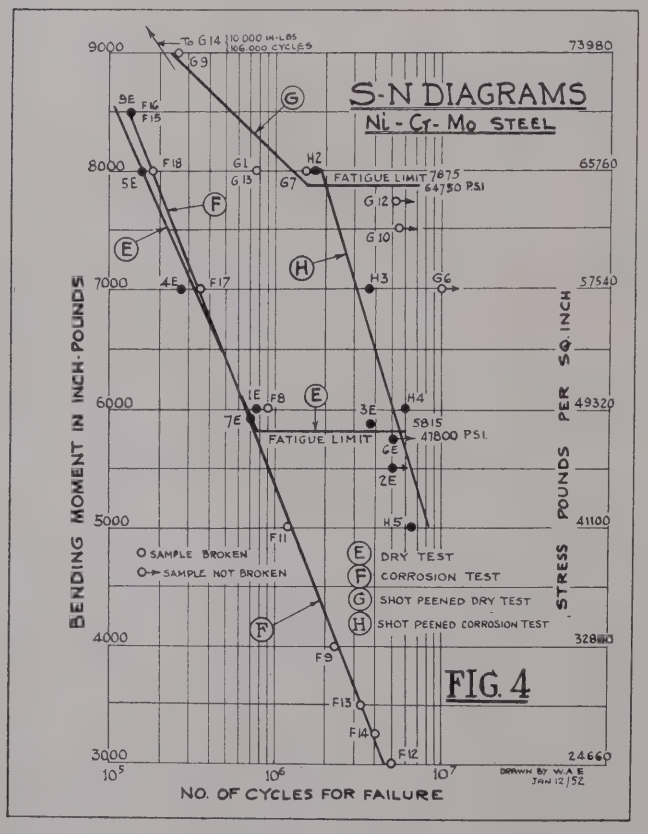

Figure 4.

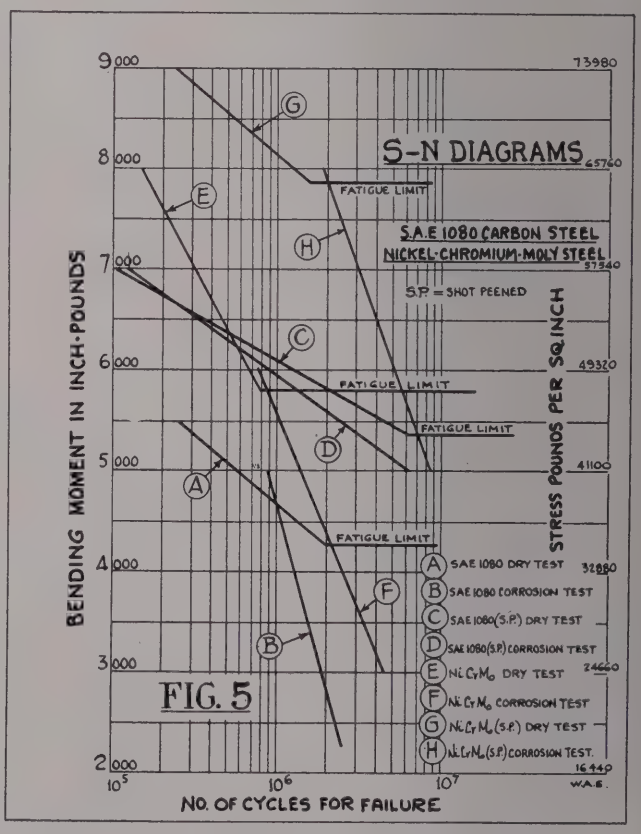
S-N DIAGRAMS
S.A.E 1080 CARBON STEEL
NICKEL-CHROMIUM-MOLY STEEL
S.P. = SHOT PEENED
FATIGUE LIMIT
BENDING MOMENT IN INCH-POUNDS
STRESS POUNDS PER SQ.INCH
NO. OF CYCLES FOR FAILURE
A SAE 1080 DRY TEST
B SAE 1080 CORROSION TEST
C SAE 1080 (S.P.) DRY TEST
D SAE 1080 (S.P.) CORROSION TEST
E Ni Cr Mo DRY TEST
F Ni Cr Mo CORROSION TEST
G Ni Cr Mo (S.P.) DRY TEST
H Ni Cr Mo (S.P.) CORROSION TEST
FIG. 5

Figure 5.

All *S-N* relationships presented in a semi-logarithmic scale are fairly regular and, for the working ranges we are interested in, could be drawn as straight lines without much deviation from the actually determined points.

The fatigue limit for the dry condition was determined for 5×10^6 cycles, and in some cases 10×10^6 and 15×10^6 tests were performed.

The ranges of stress used can be seen from Figures 3, 4, and 5.

A graphical summary of the results obtained is given in Figure 5,

TABLE III.— FATIGUE TESTS OF SAE 1080 CARBON STEEL, AS ROLLED, UNDER DRY CONDITIONS

SAMPLE NO.	BENDING MOMENT (inch-pounds)	MAXIMUM STRESS, p.s.i. (edge A and flat B)	NO. OF CYCLES TO FAILURE × 10^3
A15	6,000	49,320	149
A5	5,500	45,210	236
A1	5,000	41,100	628
A2	4,500	36,990	1,432
A7	4,500	36,990	1,241
A11	4,375	35,862	1,621
A12	4,312.5	35,348	1,212
A10	4,312.5	35,348	1,272
A13	4,275	35,140	1,900
A8	4,250	34,935	5,100*
A14	4,250	34,935	5,100*
A6	4,000	32,880	10,000*

*Sample unbroken.

All samples broke in free gauge length. The nucleus of fatigue failure was observed on edge *A* and octagonal flat *B* (see Ref. 1).

Typical dual-type fractures were obtained, with two distinct zones, namely:

(1) A smooth matte surface (M1) resulting from a crack initiated at the sharp root radius and progressing toward the internal water hole was battered smooth by the repeated opening and closing of the crack taking place during each cycle.

(2) A rough crystalline surface (X2) shows a sudden fracture resulting from overstressing.

TABLE IV.— FATIGUE TESTS OF SAE 1080 CARBON STEEL AS ROLLED, UNDER WATER CORROSION CONDITIONS

SAMPLE NO.	BENDING MOMENT, (inch-pounds)	MAXIMUM STRESS, p.s.i. (edge A and flat B)	NUMBER OF CYCLES TO FAILURE x 10^3
B13	5,000	41,100	819
B12	4,500	36,990	1,167
B2	4,250	34,935	1,288
B4	4,000	32,880	1,445
B5	3,500	28,770	1,527
B6	3,000	24,660	2,227
B7	2,750	22,605	1,956
B8	2,500	20,550	1,990
B10	2,500	20,550	2,289
B9	2,250	18,495	2,500
B14	2,000	16,440	Sample unbroken

Nucleus of corrosion fatigue failure was observed on edge *A* and octagonal flat *B*. Also, samples *B12* and *B8* show nucleus on quarter flat *C*, where extensive corrosion took place.

Typical dual-type fatigue fractures were obtained. A smooth matte surface (M1) and rough crystalline surface (X2).

Samples *B2*, *B4*, and *B5* were pre-corroded for 50 hours Ottawa tap-water.

TABLE V.— FATIGUE TESTS OF SAE 1080 CARBON STEEL, SHOT-PEENED, UNDER DRY CONDITIONS

SAMPLE NO.	BENDING MOMENT, (inch-pounds)	MAXIMUM STRESS, p.s.i. (edge A and flat B)	NUMBER OF CYCLES TO FAILURE x 10^3
C-10	7,000	57,540	104
C-11	6,000	49,320	962
C14	5,750	47,265	2,621
C13	5,500	45,210	4,792
C16	5,375	44,180	15,224*
C15	5,250	43,155	10,451*
C12	5,000	41,100	7,636*

*Sample unbroken.

All samples broke in the free gauge length, with the exception of samples *C10* and *C13*, which broke in adaptors. Nucleus of fatigue was observed on octagonal flat *B* on sample *C10* and also on quarter flat *C*. Typical dual-type fatigue fractures, with a smooth matte surface (M1) and a rough crystalline surface (X2), were obtained.

TABLE VI.— FATIGUE TESTS OF SAE 1080 CARBON STEEL, SHOT-PEENED, UNDER WATER CORROSION CONDITIONS

SAMPLE NO.	BENDING MOMENT, (inch-pounds)	MAXIMUM STRESS, p.s.i. (edge A and flat B)	NUMBER OF CYCLES TO FAILURE x 10^3
D3	7,000	57,540	137
D6	6,500	53,430	292
D2	6,000	49,320	928
D4	6,000	49,320	964
D5	5,500	45,210	2,675
D7	5,250	43,155	2,931
D8	5,125	42,127.5	3,600
D1	5,000	41,100	8,335
D9	5,000	41,100	5,169

GEBERAL REMARKS ON ABOVE TESTS

All samples broke in the free gauge length with the exception of the following samples: *D2*, which broke in right adaptor; *D6*, in left adaptor; *D7*, at the edge of collet; *D3*, in gauge length but outside of corroded section.

Nucleus of fatigue was observed on octagonal flat *B*. Typical dual-type fatigue fractures, M1 and X2, were obtained.

TABLE VII.— FATIGUE TESTS OF Ni-Cr-Mo STEEL, AS ROLLED, UNDER DRY CONDITIONS

SAMPLE NO.	BENDING MOMENT, (inch-pounds)	MAXIMUM STRESS, p.s.i. (edge A and flat B)	NUMBER OF CYCLES TO FAILURE x 10^3
E9	8 500	69,870	138
E5	8,000	65,760	154
E4	7,000	57,540	262
E1	6,000	49,320	779
E7	5,930	48,745	700
E3	5,875	48,293	3,820
E6	5,750	47,265	5,123*
E2	5,500	45,210	5,200*

*Sample unbroken.

GENERAL REMARKS ON ABOVE TESTS

All samples broke in the free gauge length. Nucleus of fatigue was on the octagonal flat *B*. Dual-type fatigue fractures, (M1) and MX1 (matte crystalline fracture), were obtained; shape of fractures irregular.

TABLE VIII.— FATIGUE TESTS ON Ni-Cr-Mo STEEL, AS ROLLED, UNDER WATER CORROSION CONDITIONS

SAMPLE NO.	BENDING MOMENT, (inch-pounds)	MAXIMUM STRESS, p.s.i. (edge A and flat B)	NUMBER OF CYCLES TO FAILURE x 10^3
F15	8,500	69,870	139
F16	8,500	69,870	139
F18	8,000	65,760	175
F17	7,000	57,540	352
F8	6,000	49,320	909
F10	6,000	49 320	532
F11	5,000	41,100	1,121
F9	4,000	32,880	2,423
F13	3,500	28,770	3,288
F14	3,250	26,715	3,942
F12	3,000	24,660	5,212

GENERAL REMARKS ON ABOVE TESTS

All samples broke in the free gauge length, except sample *F8* which broke in the adaptor, but cracks were noticed in the gauge length. Nucleus of fatigue was noticeable on the edge and small flat. Dual-type fatigue fractures, M1 and MX1, were obtained. Shapes of fractures, irregular.

TABLE IX.— FATIGUE TESTS OF Ni-Cr-Mo STEEL, SHOT-PEENED, UNDER DRY CONDITIONS

SAMPLE NO.	BENDING MOMENT, (inch-pounds)	MAXIMUM STRESS, p.s.i. (edge A and flat B)	NUMBER OF CYCLES TO FAILURE x 10^3
G9	9,000	73,980	269
G1	8,000	65,760	774
G7	8,000	65,760	1,712
G13	8,000	65,760	695
G8	7,750	63,705	1,528
G12	7,750	63,705	5,100*
G10	7,500	61,650	5,640*
G6	7,000	57,540	10,171*

*Sample unbroken.

All samples broke in the free gauge length. Nucleus of fatigue was observed on octagonal flat. Typical dual-type fatigue fractures, M1 and MX1, were obtained.

TABLE X.— FATIGUE TESTS OF Ni-Cr-Mo STEEL, SHOT-PEENED, UNDER WATER CORROSION CONDITIONS

SAMPLE NO.	BENDING MOMENT, (inch-pounds)	MAXIMUM STRESS, p.s.i. (edge A and flat B)	NUMBER OF CYCLES TO FAILURE x 10^3
H2	8 000	65,760	1,757
H3	7,000	57,540	3,677
H4	6,000	49,320	6,017
H5	5,000	41,100	6,548
H6	4,000	32,880	7,330

GENERAL REMARKS ON ABOVE TESTS

All samples broke in the free gauge length. Nucleus of fatigue was observed on octagonal flat. Typical dual-type fatigue fractures, M1 and MX1, were obtained.

which presents a general view of the problem and, also, information on the absolute and the relative data for the eight conditions investigated.

The numerical results of the simulated service tests, presented graphically in the form of *S-N* curves in Figures 3, 4, and 5, are given in Tables III to X.

Symbols Key

The following symbols, used for the classification of the structure of the fractured specimen surfaces, are taken from the Mines Branch *Tables for the Classification of Fractures*:

M1 — Matte fracture, fine grain

M2, M3 — Matte fracture, medium and coarse grain

XI — Crystalline fracture, fine bright grain

X2, X3 — Crystalline fracture, medium and coarse bright grain

MX — Matte-crystalline fracture

METALLOGRAPHIC EXAMINATION AND THE MECHANISM OF FAILURE IN FATIGUE

All service failures of drill steel are attributed to some type of fatigue failure. The main mechanical properties that should be used in analysing the field performance of drill steel are its fatigue limit under dry conditions, and the endurance strength under water-corrosion conditions.

The simplified mechanism of fatigue failure of drill steel under alternating stress could be described as follows: When the magnitude of an externally applied static and dynamic load reaches a value which will produce, along various crystal planes, a shear stress exceeding the elastic limit in shear, a local plastic deformation takes place. This local initial deformation results in so-called strain hardening (cold work). If the shear stresses when resolved along preferred planes having the least resistance to slip are not large, further slip is prevented and the metal does not show local changes due to cold work. Large alternating shear stresses will produce progressive cold work. If this cold work is allowed to exceed a certain critical value it will produce local structure discontinuities, resulting in small cracks. During alternating loading, these small cracks start to grow in number and size and, if they reach a critical level, the nuclei of fatigue failure are formed and the metal fails.

The surface of mining drill-rods in the as-rolled condition is covered with a large number of local pits, sharp notches, and other irregularities. These surface imperfections are caused by: closed, but unwelded, surface cracks in ingots and billets elongated during rolling; rolled-in splash and oxides on the ingot surface; elongated slivers of metal adhering to the billet; rolling laps; other miscellaneous conditions.

Typical examples of surface contours are shown in Figures 6 and 8.

These surface imperfections introduce large local stress concentrations (7) which may be up to two to three times larger than the nominal stresses obtained from calculations or direct surface measurements. The magnitude of these stress concentrations depends on the sharpness of the root at the bottom of the notch.

A large number of stress concentrations caused by surface imperfections produce, in the neighbourhood of their roots, local plastic deformations which might relieve these concentrations to a certain extent, but, if the amount of plastic flow exceeds a critical value, a nucleus of fatigue will be formed.

In drill rods, the maximum stresses appear at the outside surface and it is clear that the nucleus of fatigue failure should form there.

The results of our simulated service tests are presented in the form of eight *S-N* curves, shown in Figure 5. In order to complete this picture it was considered desirable to reveal, by microscopic examination, the mechanism of fatigue failure of drill steel (see Figures 6 to 11).

Figure 6 shows a longitudinal contour of the *SAE 1080* drill rod as-rolled, unetched. The sample (*A13*), under dry fatigue conditions, broke at a bending moment of ±4,275 inch-pounds (35,140 p.s.i.) after 1.9×10^6 cycles. Visible irregular surface imperfections with sharp, deep notches introduce local stress concentrations at the bottom of the roots, where the nucleus of fatigue could be formed, preceded by a large local plastic deformation. The right part of the contour (Figure 6) shows that the sample has a relatively smooth surface; the centre and the left part of the contour represents the most irregular surface conditions which could be located. Eight longitudinal sections approximately 1½ inches long were examined microscopically, and the only crack detected was the one which caused the failure of the sample investigated.

Figure 7, similar to Figure 6 but etched in 2 per cent nital, shows a decarburized layer 0.006 inch deep.

Figure 8 shows the longitudinal contour of the *SAE 1080* drill rod, sample No. D*9*, as-rolled and shot peened, which broke after $5{,}169 \times 10^6$ cycles at a bending moment of ±5,000 inch-pounds (±41.000 p.s.i.) under water-corrosion conditions.

Visible initial surface imperfections, with sharp and deep-rooted corrosion cracks originating at the bottoms of the roots through the combined effect of local stress con-

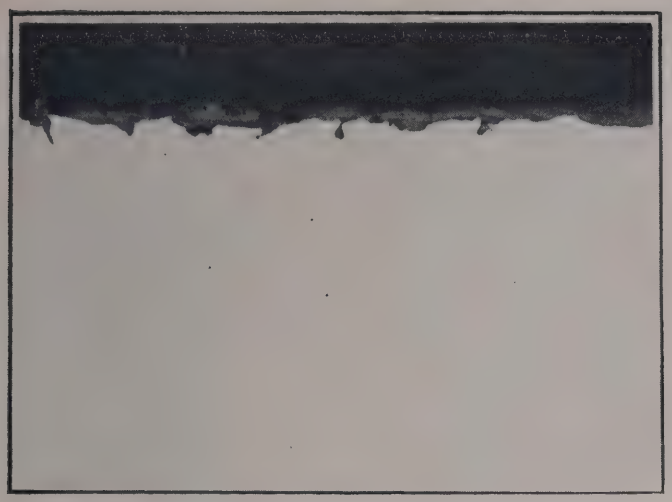

Figure 6.—Longitudinal contour of SAE 1080 drill rod, as rolled, unetched. ×100.

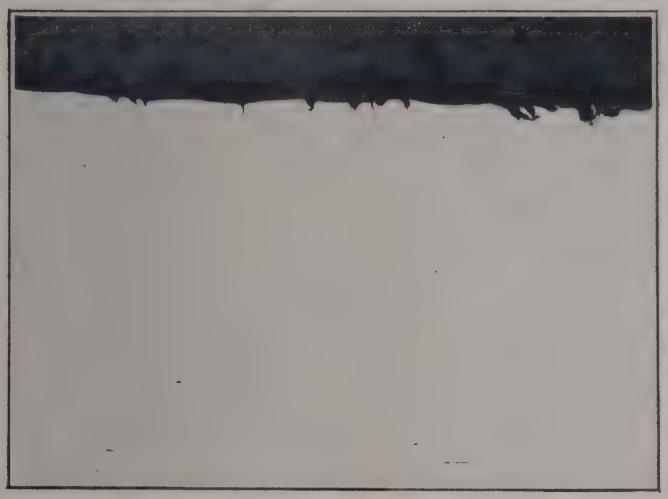

Figure 8.—Longitudinal contour of SAE 1080 drill rod, as rolled and shot peened. ×100.

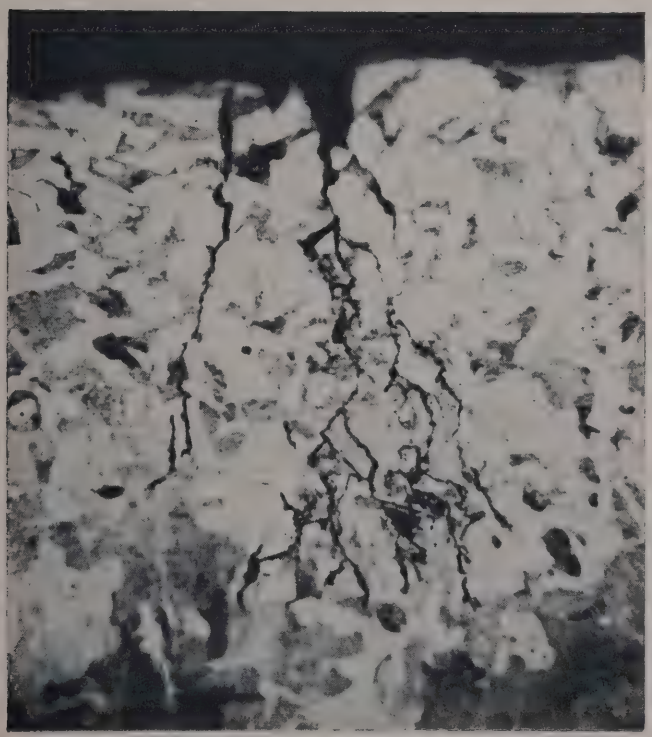

Figure 9.—Same sample as in Figure 8 but etched in 2 per cent nital. ×500.

Figure 7.—Similar to Figure 6 but etched in 2 per cent nital. ×100.

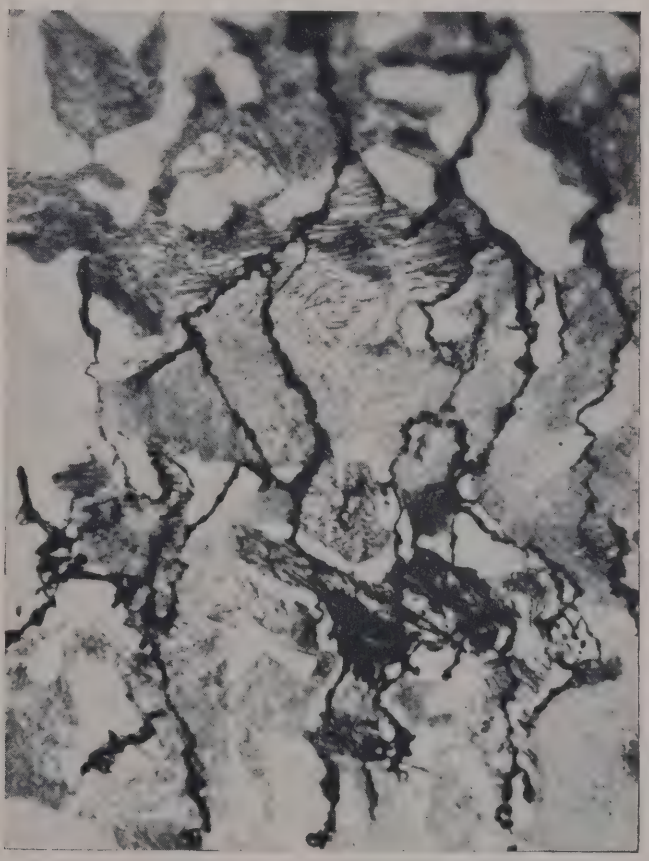

Figure 10.—Middle portion of sample shown in Figure 9.

centrations and corrosion, are potential causes of corrosion-fatigue failure. Figure 9 of the same sample as in Figure 8 but etched in 2 per cent nital, shows two corrosion fatigue cracks originating in a similar way to the numerous cracks shown in Figure 8. The general dircetion of the propagation of these cracks is perpendicular to the normal bending stresses.

Figure 10 shows the middle portion of the sample. Numerous fatigue cracks, whose local directional changes were caused by variations in the cohesive strength of different crystal planes of pearlite and ferrite are visible in this figure.

Figure 11 shows the longitudinal section of the Ni-Cr-Mo drill rod, as-rolled. Sample *F8* broke after 909,000 cycles and at a bending moment of 6,000 inch-pounds (±49,320 p.s.i.), under water corrosion.

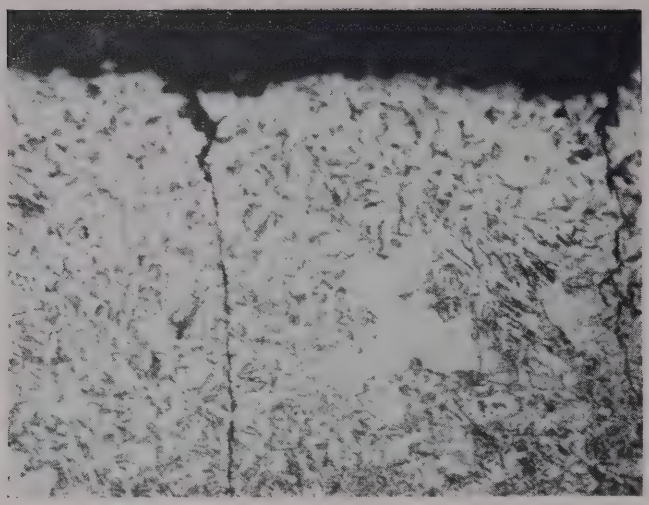

Figure 11.—Longitudinal section of Ni-Cr-Mo drill rod, as rolled. ×500.

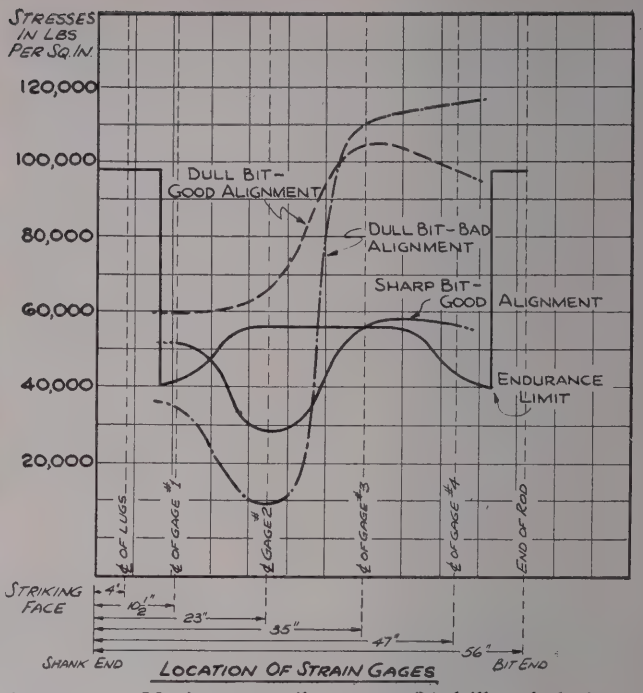

Figure 12.—Maximum tensile stresses in drill rod during drilling operation.

Initial surface imperfections, with two sharp and deep-rooted corrosion cracks, are visible. The cracks are perpendicular to the maximum normal stress. Because of the greater homogeneity of the cohesive strength of the Ni-Cr-Mo steel, the cracks are much more regular than those in *SAE 1080* shown in Figures 9 and 10.

Discussion and Conclusions

Conclusions on the performance of the steels investigated are presented in Tables XI, XII, and XIII. In Table XI, the overall notch sensitivity of drill rods under eight different conditions from *A* to *H* is given. From this table and the corresponding diagrams (Figure 3, 4, 5), the effect of water corrosion and shot-peening may be summarized as follows. For *SAE 1080* steel, as-rolled, in the dry condition, the notch sensitivity of 1.86 increases under water corrosion to 2.8 for 2 million cycles and to 12.7 for 5 million cycles; after shot-peening, the dry notch sensitivity decreases to 1.48 and, after shot peening under water-corrosion, to 1.43 (1.55). This comparison shows that shot-peening substantially reduces the notch sensitivity of both drill steels and, therefore, very effectively increases its field performance under dry and water corrosion conditions.

A clear and simple tabulation of the characteristics of the laboratory and field performance of drill rods is given in Tables XII and XIII. In Table XII are given the number of cycles to failure for different stress levels. In Table XIII, the maximum stress required for failure at different cycles is tabulated. Practical conclusions could be drawn from these tables as to the effect of water-corrosion on the field performance of drill steel, and as to the beneficial effect of shot peening.

The previously introduced simplified mechanism of fatigue failure, supported by metallographic examination, gives useful information on the causes of failure of drill steel.

From the *S-N* diagrams shown in Figure 5, several deductions could be drawn. A few typical remarks are given below. The highest endurance strength under dry and water-corrosion conditions is obtained using Ni-Cr-Mo steel after shot peening (see curves *G* and *H*) (8).

From the relative position of curves *E* and *F*, it is obvious that the endurance strengths are similar. To a certain extent, the same could be said about curves *C* and *D* drawn for plain carbon shot-peened steel.

Table XI.—Overall Notch Sensitivity of Mining Drill Rods

Surface and Condition of Test	Overall Notch Sensitivity*	
	50% U.S. / FL. or E.S.	0.01% P.S. / FL. or E.S.
SAE 1080 Steel:		
A. —As rolled	1.86	1.68
Dry fatigue		
B. —As rolled	2.8 (2 million cycles)	2.53
Corrosion fatigue	12.7 (5 million cycles)	11.5
C. —Shot peened	1.48	1.34
Dry fatigue		
D. —Shot peened	1.43 (2 million cycles)	1.29
Corrosion fatigue	1.55 (5 million cycles)	
Ni-Cr-Mo Steel:		
E. —As rolled	1.75	1.52
Dry fatigue		
F. —As rolled	2.32 (2 million cycles)	2.01
Corrosion fatigue	3.59 (5 million cycles)	3.12
G. —Shot peened	1.29	1.13
Dry fatigue		
H. —Shot peened	1.3 (2 million cycles)	1.12
Corrosion fatigue	1.7 (5 million cycles)	1.47

U.S.— Ultimate tensile strength
F.L.— Fatigue limit
*Estimated strength reduction factor

P.S.— Proof stress
E.S.— Endurance strength

By examining the relative positions of curves *A* and *B* for *SAE 1080* plain carbon steel, the beneficial effect of water cooling can be observed for a high stress level below a million cycles. A similar tendency is also shown by curves *G* and *H* for Ni-Cr-Mo steels.

Since the *S-N* relationships shown in Figure 5 are linear, the approximate evaluation of the dynamic characteristic of drill steel by means of the endurance curve can be satisfactorily determined by three points. The relative endurance strength of different steels could thus be quickly ascertained.

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* * *

Contributed Discussion

Mr. F. R. Anderson (1): The proposed new method of testing mining drill rods, developed by Dr. T. W. Wlodek, is very simple and could easily replace the presently used old method, *i.e.*, the so-called block test.

At the Gardner-Denver Research Laboratory we have been attempting the measurement of stresses occurring in drill steel during operations. We have used a Hathaway recording oscillograph which permits measurement of stress fluctuation by means of *SR-4* gauges. The results of our measurements are shown on the attached graph (Figure 12).

Our stress analyses were made on a 4-foot drilling length of one-inch quarter-octagonal steel, at one end of which was assembled a 2-inch Timken steel bit. The drill steel was operated in a 3½-inch Gardner-Denver tappet-type *D89* drifter for plain stoper shanks, at 80 p.s.i. air pressure.

During these stress measurements, the bit was engaged in actual drilling operations in a hard rock. While stress measurements were being taken, the drill rod was rotating. The *SR-4* electrical strain gauges were fixed on the large flats in places marked on the graph. The distribution of maximum tensile stresses, as determined by this test, are given on the graph.

From our measurements we see that the drill steel is exposed to a wave of tension-compression direct stress loading. It is also to be expected that the tension-compression direct stress wave is superimposed by vibrating bending stresses.

In the proposed testing method, the drill rod fails under equal tension-compression stress, and in actual drilling operations our stress analysis shows that the tension-compression type of loading exists and therefore we can assume that, in this laboratory testing method, working conditions are simulated.

(1) Chief Metallurgist, Gardner-Denver Company, Denver, Colorado.

Table XII.—Number of Cycles for Given Stresses

In.-lb.	7,000	6,000	5,000	4,000	3,000
P.s.i.	57,540	49,320	41,100	32,880	24,600
A	(20,000)	110,000	0.63×10^6	—	—
B	(320,000)	(500,000)	0.82×10^6	1.4×10^6	1.9×10^6
C	104,000	1.4×10^6	—	—	—
D	137,000	0.928×10^6	6.75×10^6	(45×10^6)	(333×10^6)
E	325,000	675,000	—	—	—
F	352,000	670,000	1.3×10^6	2.5×10^6	4.7×10^6
G	—	—	—	—	—
H	3.1×10^6	5.1×10^6	8.5×10^6	(14×10^6)	(22×10^6)
F/B	(1.10)	(1.24)	1.59	1.79	2.47
F/D	2.57	0.722	0.193	(0.0556)	(0.014)
H/B	(9.69)	(10.2)	10.4	(10.0)	(11.6)
H/D	22.6	5.5	1.26	(0.311)	(0.0661)

Note—The figures given in brackets were obtained by extrapolation outside of the testing range.

Table XIII.—Stresses for Given Number of Cycles

Cycles×1,000	100	500	1,000	2,000	5,000
A	49,700	42,900	38,500	35,000	35,000
B	(74,000)	(53,400)	38,500	23,400	(5,140)
C	57,500	52,700	50,400	48,000	45,200
D	58,300	51,900	48,800	45,900	42,300
E	(69,900)	52,400	47,800	47,800	47,800
F	73,400	51,900	44,300	35,400	23,700
G	(78,000)	70,000	67,000	64,700	64,700
H	(111,000)	(87,300)	(75,500)	64,700	49,500
F/B	(0.992)	(0.972)	1.15	1.51	(4.61)
F/D	1.26	1.00	0.908	0.771	0.560
H/B	(1.50)	1.63	(1.96)	2.76	(9.63)
H/D	(1.9)	1.68	(1.55)	1.41	1.17

Note: The figures given in brackets were obtained by extrapolation outside of the testing range.

Mr. Gunnar Lilljekvist (2) : We have read with great interest Dr. Wlodek's paper.

The fatigue testing machine and the method used should give particularly valuable results regarding the fatigue strength of the surface layer of drill steel and also on the soft transition between the unaffected bar and portions which have been heated for forging or brazing. Of course, only the effect of bending stresses is obtained and not of longitudinal vibrations as during usual drilling.

The testing method seems, however, to give a number of interesting data regarding the steel and it would be valuable if we could get some idea regarding the cost and delivery possibilities for such a machine.

With the suggested clamps, can the risk of breakage be avoided in the clamping device? We assume that clamps are available for both round and hexagon steel.

The arrangements for corrosion fatigue testing are very interesting for us, especially with reference to stainless steel and similar shaft material.

(2) Director of Laboratories, Avesta Iron and Steel Works, Sweden.

Author's Reply

Mr. Gunnar Lilljekvist and Mr. F. R. Anderson have pointed out some interesting facts about the advantages of the method used and results obtained.

At present, we are using our method of fatigue testing of mining drill rods in research on the dynamic characteristic of the so-called 'Metallurgical Notch' *i.e.,* a soft transition zone between the unaffected bars and the heated portion of the drill rod.

The danger of breaking the sample in the grips is virtually eliminated by shot peening the gripped portions and using bakelite and brass shims with a suitable lubricant.

Mr. Anderson introduced a very interesting example of maximum values and the character of the stress pattern to which the drill set is exposed. This is an important contribution to our research. From the graphical results of measurements of the distribution of dynamic stresses on a drill rod during actual drilling operations shown by Mr. Anderson, a number of conclusions could be drawn. I would like to stress the large magnitude of dynamic tension stresses observed at the bit end. Having similar stress data for typical mining and operating conditions, it would be possible to evaluate from our *S-N* relations the number of cycles (blows) to failure and the time of drilling.

The author wishes to take this opportunity to thank Messrs. Anderson and Lilljekvist for the interesting discussion submitted on the foregoing paper.

The Use of Ceramic Tests in the Correlation of Strata

*By A. S. DAWSON**

(*Ottawa General Meeting, January,* 1952)

(*Transactions, Volume LV,* 1952, pp. 291-295)

Introduction

THE PURPOSE of this paper is to describe the results of a preliminary investigation of a method thought to be useful in interpreting geological structures in unmetamorphosed sedimentary rocks. The writer is not aware of the method having been described or investigated previously, although no thorough search of the literature has been made.

The method to be described is based on the assumption that, in argillaceous sedimentary rocks, certain ceramic properties are diagnostic for certain beds within areas of variable limits.

Two ceramic properties have been investigated: pyrometric cone equivalent (P.C.E.), and bloating characteristics of shales. The enquiry has been directed toward testing the usefulness of these properties in determining the following:

A.—The correlation of shale beds and, by inference, of coal seams, sandstone beds, or other such structures associated with them.

B.—The location of contacts between marine and non-marine shales where lithological, palaeontological, or other evidence is lacking.

Limitations of time and facilities have prevented a thorough investigation of this problem. The results given in the present paper are preliminary only. It is hoped, however, that they will occasion discussion, or even lead to more thorough investigation of the method by individuals or organizations.

There are, admittedly, various limitations to the proposed method. It is thought to show some promise when used in conjunction with the well-known and established methods used in stratigraphic correlation and, more particularly, when the conventional methods are not applicable. The field of its usefulness, both geographically and with reference to the geological time-scale, is not known.

*Field Manager, Western Potash Corporation, Limited. Unity. Sask.

Acknowledgments

The writer is indebted to the following: Mr. J. G. Matthews, Mines Branch, Ottawa, for early field assistance and for determinations of bloating characteristics; Mr. Ian F. Wright, Mines Branch, Ottawa, for many P.C.E. determinations; Mr. C. S. Parsons, Mines Branch, Ottawa, for authorizing the laboratory work; Mr. M. B. B. Crockford, Petroleum and Natural Gas Conservation Board of Alberta, Calgary, for helpful advice and for making oilwell samples available; Mr. C. S. Dewis, The Canmore Mines, Ltd., Canmore, for allowing and assisting the writer to take samples on that Company's property; Mr. F. D. Shepherd, Mid-West Metal Mining Association, Winnipeg, for helpful advice and criticism; Mr. G. I. Samson, Canadian Pacific Railway Company, Calgary, for field and stenographic assistance; and others too numerous to mention.

Applications of the Method

The method is believed to have both academic and practical applications. Examples of possible practical applications are as follows:

A.—In correlating coal seams in closely folded and faulted areas. The Kootenay coal measures of western Alberta and eastern British Columbia represent a field in which the proposed method might be useful. A recent paper by Newmarch (1) described a method of correlating Kootenay coals by means of spectrochemical analyses of the ash. Newmarch's paper also included an excellent summary of existing methods of correlating coal seams. It is suggested that the present method might prove useful in conjunction with that described by Newmarch, or in areas where that method does not prove of diagnostic value. The extent of its usefulness is, however, not known.

B.—As an aid to determining structure in oil exploration. If otherwise uncorrelatable shale beds can be correlated by this method, then new marker beds will be available which might be useful in determining structure by drilling or on surface. If the method is applicable to structure drilling, these 'marker beds' might well occur at much shallower horizons than those conventionally used. Where correlations suggested by electro-logs are indefinite, this method might be used as a check.

(1) For references, see end of paper.

Description of Preliminary Results

In the testing done to date, the purpose has been to take samples from structures whose interpretation was previously known or thought probable. Samples of shale were taken in the following areas and geological horizons in Alberta: the Redcliff area (Foremost formation), the Canmore area (Kootenay formation), and the Lethbridge area (Bearpaw and Oldman formations). These areas were chosen at random and mainly because of accessibility or convenience.

Oilwell samples were also obtained and tested from the Stettler and Conrad areas. These samples each represented ten feet of cuttings, but had been reduced for storage to only one or two ounces in size. They were also contaminated with foreign material caved from higher elevations in the wells. These features of the samples possibly account for the fact that their testing did not give results of correlative value. The other samples, which gave the more positive results described below, were larger, represented much smaller vertical intervals, and were taken with considerably more care in view

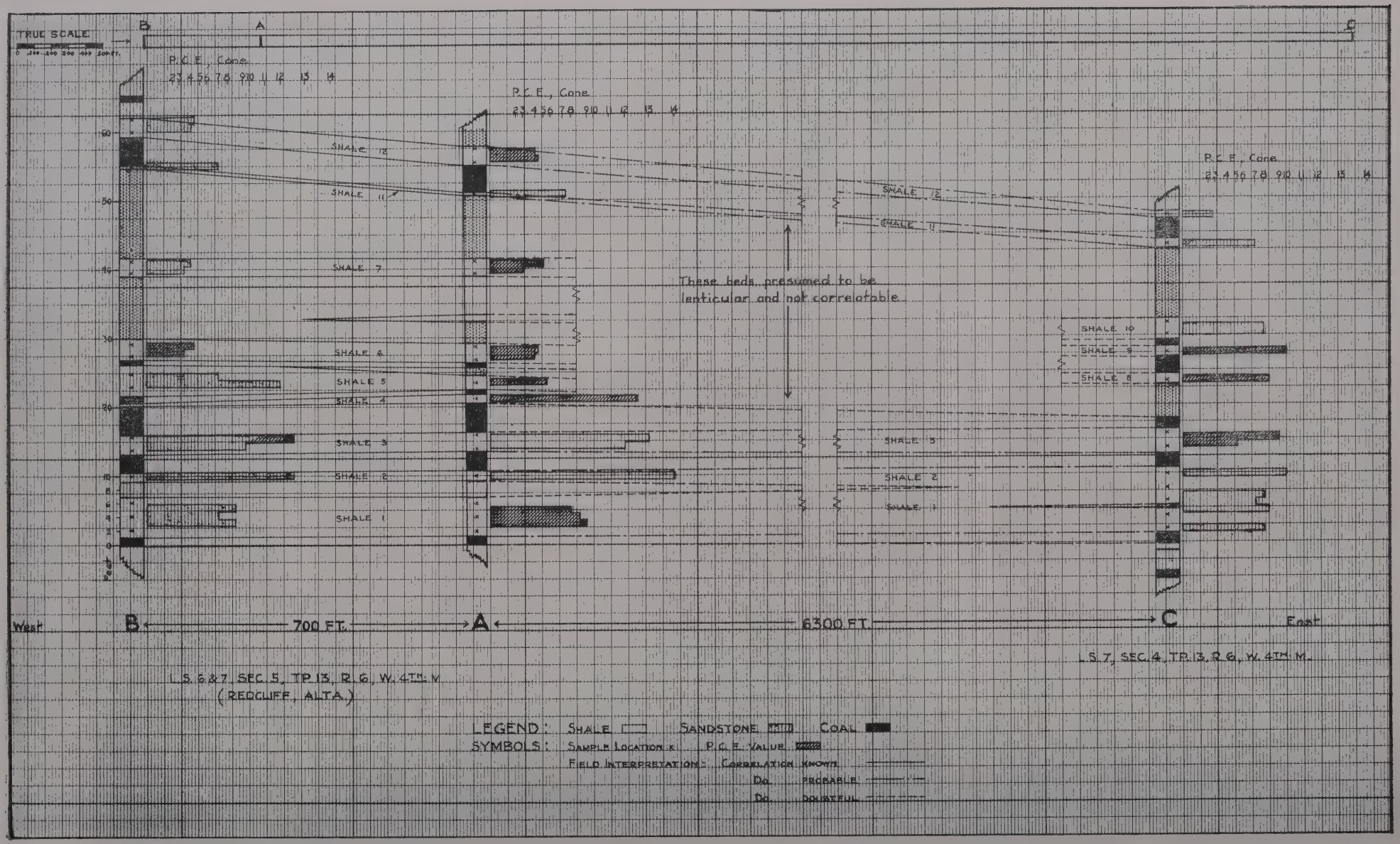

Figure 1.—Chart showing three stratigraphic sections at Redcliff and correlations based on P.C.E. values.

of the particular purpose of the test.

Correlation of Shale Beds

Redcliff Area

The Oldman and Foremost formations in this area have been described by Crockford (2). The three stratigraphic sections sampled are shown diagramatically in Figure 1. Fig. 3, p. 508 of Crockford's paper, shows twelve stratigraphic sections measured along South Saskatchewan river. That shown second from the right corresponds roughly to sections *A* and *B*, and that at the right to section *C* of the accompanying Figure 1. The three sampled sections are in the Foremost formation, 50 to 60 feet high, and their bottoms are about ten feet below Crockford's stratigraphic datum.

The following comment applies to Figure 1: The three sections sampled are shown in their true relative horizontal and vertical scales at the top of the figure; the height of the sections is about 8.6 per cent of the distance between *A* and *B*, and about 0.9 per cent of the distance between *A* and *C*. The main part of the figure is diagrammatic.

The correlation between sections *A* and *B*, 700 feet apart, based on P.C.E. values of the shale samples, is considered so definite that little comment is required. The two patterns obtained from the samples are so nearly identical that no other interpretation of the results is possible. This correlation was previously known, it being possible in the field to walk out the beds concerned.

The correlation between sections *A* and *C*, 6,300 feet apart, must be based on probabilities rather than on known facts, since the beds cannot be walked out between the sections. The correlations shown in Figure 1, based on P.C.E. values, are the same as those thought probable by earlier field investigators.

In sections A and C, the P.C.E. values for shales 11 and 12 correspond closely. The somewhat lower value for shale 12, section *C*, is due to the shale being slightly calcareous at this point, whereas in sections *A* and *B* it is not.

Values for shales 8, 9, and 10, section *C*, do not correspond to those for shales 4, 5, 6, and 7, section *A*. Field relations also indicate that these beds are probably not correlatable.

Correlation of shale 3 is based on two points: this bed gave the second highest value in its respective section; the lower part of the bed is appreciably less refractory than the upper part, a pattern not duplicated by any of the other beds.

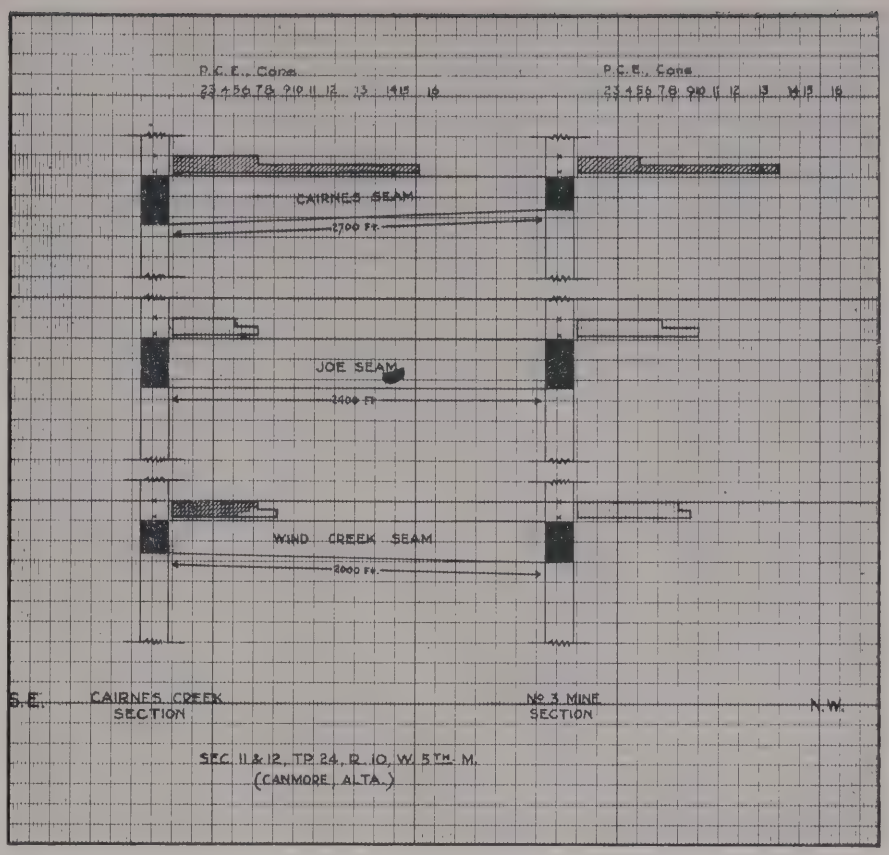
P.C.E. Cone
2 3 4 5 6 7 8 9 10 11 12 13 14 15 16
P.C.E. Cone
2 3 4 5 6 7 8 9 10 11 12 13 14 15 16
CAIRNES SEAM
2700 Ft.
JOE SEAM
2400 Ft.
WIND CREEK SEAM
2000 Ft.
S.E.
CAIRNES CREEK SECTION
No. 3 MINE SECTION
N.W.
SEC. 11 & 12, TP. 24, R. 10, W. 5TH. M.
(CANMORE, ALTA.)

Figure 2.—Chart showing correlations of coal seams near Canmore, based on P.C.E. values.

In shale 2, the single value determined is, in each case, the highest in its respective section.

A small liberty was taken in showing the values for shale 1. This was done to show clearly the close similarity of the results after ignoring the presumed pinching out of thin sandstone beds.

The writer's belief, shared by others, is that a probable correlation can be made between section *A* and *C*, based on P.C.E. values alone.

Canmore Area

Samples taken in this area give a preliminary indication of the applicability of the method to the correlation of Kootenay coal seams. Two localities on each of three seams were sampled. Owing to lack of large exposures, it was possible to obtain only two samples, at two-foot vertical intervals, above the coal at each locality. The seams are exposed in test pits in the southern part of the property of The Canmore Mines, Limited.

Results of P.C.E. tests on the shale samples are shown in Figure 2. It will be seen that the relatively refractory shale immediately above the Cairnes seam is diagnostic for that seam at the localities sampled. The two other seams can then be correlated by super-position, if not by their P.C.E. values alone. A larger number of samples might have resulted in more distinctive, or diagnostic, patterns. The correlations shown were previously thought probable by the mine operators and other investigators.

The Canmore shales are not good natural bloaters. This perhaps accounts for the fact that the numerical results of bloating tests are erratic and of little diagnostic value. However, the qualitative results from heating the samples for three minutes at 2,300°F. are of some interest, as indicated in Table I.

The four samples associated with the Wind Creek seam are all well bloated. In addition, they give light-grey products in contrast to the nearly black, high-carbon products from the other samples. This condition seems to be diagnostic for that seam within the scope of the test.

Location of Marine-Non-marine Contacts

It is not uncommon in Alberta and elsewhere to find considerable difficulty in separating shale beds of different origin. The existence of this problem is illustrated by the following quotation:

"It should be noted that in the logs of Medicine Hat district, the

Belly River is with difficulty separated from the Pakowki, both consisting of soft, dark shales. The resemblance of the lower part of Belly River formation to the Pakowki increases toward the east" (3).

Since the above was written, great improvements in electric logs have probably gone far to eliminate such problems as encountered in oilwells. However, it is not certain that electrologs can be counted on to give a definite separation under the conditions described. Furthermore, in cases where the electrolog is not applicable, such as in surface exposures, a practical method of separation might be useful.

The thought is that such shale beds, deposited under different conditions, might react differently to ceramic tests. If so, a separation can be made.

Lethbridge Area

The literature shows some difference of opinion as to the placing of the Bearpaw-Oldman contact near Lethbridge. Russell and Landes (4) place it at the top of the uppermost coal seam. However, where an Ostrea or Corbicula bed appears above the coal, it is included in the Oldman. Link and Childerhose (5), on the other hand, put the contact at the top of a thin, brown-weathering sandstone bed which lies six to fifteen feet above the uppermost coal seam. In the present section studied, the sandstone bed (hard, sandy shale) appears to lie immediately above an Ostrea bed, so that there is here little difference in the two interpretations.

A shallow well was drilled near Lethbridge in 1950 and crossed the Bearpaw-Oldman contact. Shale samples taken every three feet were later tested for bloating characteristics and for P.C.E. Results of these tests in relation to the log of the well are illustrated in Figure 3.

The curve for P.C.E. values in this figure shows a relatively smooth outline in the upper part, which is known to relate to the marine Bearpaw shale. The outline is erratic in the lower part, made up of non-marine Oldman shale and coal. It is difficult, however, to identify the exact point where the change occurs. The curve showing results of bloating tests has a very sharp break at footage 66. This suggests to the writer that the contact is probably at footage 66, 9 feet above the sandstone bed, 11 feet above the Ostrea bed, or 26 feet above the uppermost coal seam.

There is no other direct evidence to support this contention. Investigations in the Cypress Hills area are said to have shown, from microfossil evidence, that this contact is some few feet above the customarily placed base of the Bearpaw.

TABLE I.—QUALITATIVE RESULTS OF BLOATING TESTS ON CANMORE SAMPLES
(2,300°F.—3 min.)

SAMPLE NO.	REMARKS	SEAM	SAMPLE NO.	REMARKS
6	Slight bloating	Cairnes	4	Overbloated
5	No bloating		3	Slight bloating
8	Mostly overbloated	Joe	2	Mostly overbloated
7	Slight bloating		1	Slight bloating
10	Mostly well bloated	Wind	12	Mostly well bloated
9	Mostly well bloated	Creek	11	Mostly well bloated
Cairnes Creek section (S.E.)			No. 3 Mine Section (N.W.)	

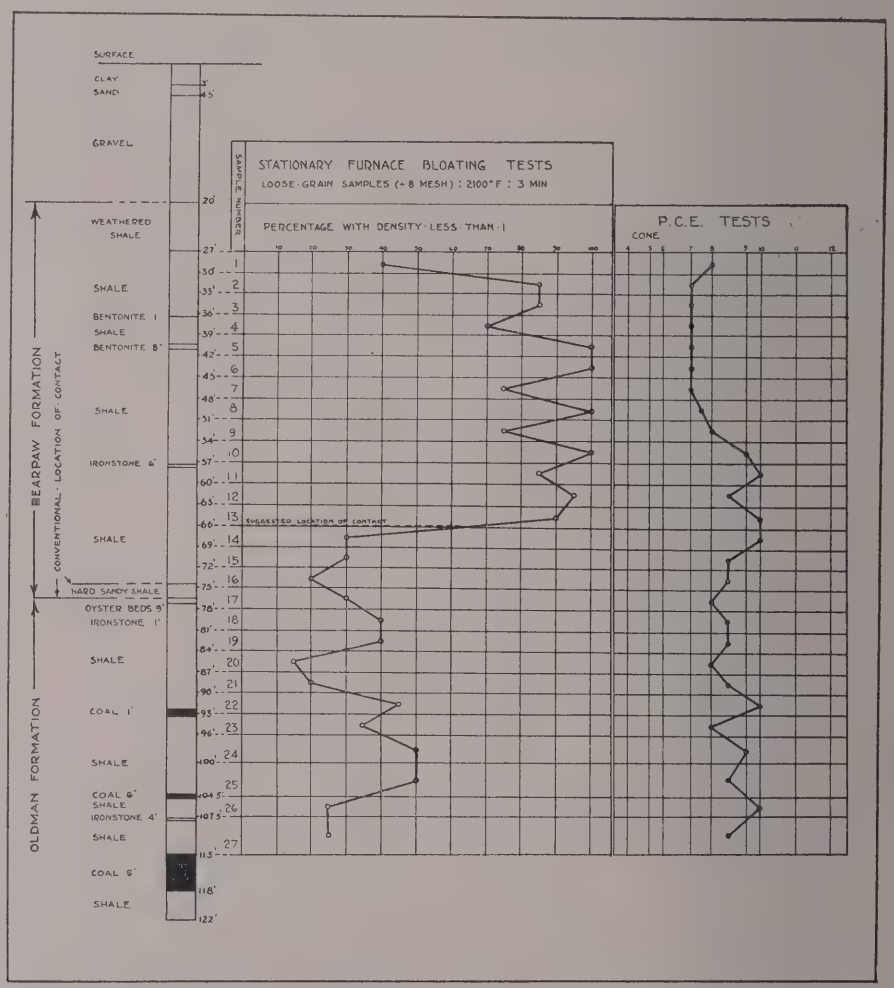

Figure 3.—Chart showing the results of P.C.E. and bloating tests in relation to the log of a well at Lethbridge.

THEORETICAL CONSIDERATIONS

Pyrometric cone equivalent (P.C.E.) is the term used to designate a property indicative of the refractoriness of a ceramic material. It replaces such unsatisfactory terms as 'fusion point' and 'deformation temperature'.

The P.C.E. of a shale is, generally speaking, a function of its chemical composition and probably also, to some extent, a function of its mineralogical composition. Composition is rather closely allied to lithology. P.C.E. tests are, therefore, a relatively simple method of measuring minor differences in composition and, presumably, in lithology.

This relationship in itself would indicate, on theoretical grounds, that the present method would supply few data of diagnostic value in correlating strata. The fact that the

method does in cases give results is difficult to explain. It seems that the amount of certain minerals and/or chemical radicals is diagnostic for some shale beds, has a marked effect on their refractoriness, and is persistent over areas of appreciable size.

Lateral variation by increase of such constituents as sand (silica) and calcium carbonate might be expected to render the same bed more refractory or less refractory, respectively. This would be especially true in the case of beds of non-marine origin. In the cases tested, however, such variations seem only to have a slight masking effect, without vitiating the results. Furthermore, these variations can be noted in both the field and the laboratory and allowances made for them in interpreting the results.

The bloating phenomenon is due to chemical reactions liberating gases at temperatures at which the material is in a pyro-plastic or semi-fused condition. It results in a light, cellular structure. The nature of the chemical reactions is not yet well understood, but they are believed by some authorities to depend primarily on mineralogical rather than chemical composition. Bloating characteristics of a shale are, therefore, an easily measured function of, probably, its mineralogical composition.

Sampling Procedure

Samples taken from natural exposures, such as those at Redeliff and Canmore, were spot samples weighing about one-half pound. They were taken at two-foot vertical intervals. Badly weathered surface material was removed to obtain relatively fresh shale. Careful notes were taken of lithology. Thickness of beds was measured by tape or hand level.

Samples from the shallow well at Lethbridge each represented three feet and consisted of cuttings. These were not appreciably reduced in size before testing.

Laboratory Procedure and Equipment

Standard procedure for determining P.C.E. has been laid down by the American Society for Testing Materials.* It is described in various textbooks on ceramic testing. A useful summary is given in a pamphlet published by the principal American maker of standard pyrometric cones (6).

*A.S.T.M. designation C24-46.

The procedure, in brief, is as follows:

(1) Crush the entire sample to minus ¼-inch and quarter to 250 gm. (0.5 lb.).

(2) Grind the reduced portion to pass a 65-mesh Tyler screen and quarter to 60 gm.

(3) Calcine the final sample in an oxidizing atmosphere for 30 min. at 1,700-1,800°F.

(4) Add an organic binder and form a test cone in a suitable metal mold.

(5) Mount test cones on a plaque of refractory material in an alternating arrangement with standard pyrometric cones. One side of each cone is mounted at an angle of 82° with the horizontal.

(6) Heat the plaque in a suitable furnace at such a rate that Cone 20 (approximately 2,800°F.), is reached in 45 minutes. The furnace need not be cooled below 1,830°F. before inserting a new plaque.

(7) When the tip of a test cone touches the plaque, note the standard pyrometric cone that most nearly corresponds to this position.

The equipment includes that for crushing and grinding, which can be done by hand if the number of samples is moderate. The furnace should be capable of reaching 2,800°F. and be fired by gas, oil, or electricity. A metal mold, preferably of steel, for making test cones, can be purchased or made in a machine shop from available designs. Plaques, also, can be purchased or made by hand from a mixture of fire clay grog and plastic fire clay. A drier of simple design is desirable.

Standard pyrometric cones may be purchased from ceramic supply firms or from The Edward Orton Jr. Ceramic Foundation, Columbus, Ohio. Their cost is nominal, thought to be between one and two cents each. Cones Nos. 1 to 20 will cover practically all shales likely to be found in a geological setting such as that of Alberta.

The P.C.E. test is a relatively simple one. Any organization already having a laboratory could install the required equipment at moderate cost. Technicians experienced along other lines can be trained in a short time in the making of this test.

The bloating test, as done for the present investigation, is very simple. First, the most suitable time and temperature for bloating are determined, by trial and error or by previous experience, for a given batch of samples. The samples, each consisting of a number of loose grains (say, plus 8-mesh, minus 4-mesh), are then placed in a furnace at the determined temperature for the determined time. For the Lethbridge samples, this was 2,100°F. for three minutes. After bloating, the percentage of each sample that floats on water is noted. This is a function of the average specific gravity and of the bloating characteristics of the sample. Some shales are not amenable to this test.

Most furnaces are suitable for both the P.C.E. and the bloating test. A pyrometer is required for the latter and is desirable for the former.

Conclusion

Within the limited scope of the present test, the suggested method gives indications of value in the correlation of strata. Specific correlation problems may exist that cannot be solved by conventional methods. In such cases, this method seems worthy of a trial, provided the structures to be correlated are associated with argillaceous beds. Purely academic studies might also be pursued along similar lines.

Before closing, the writer would like to express again his appreciation of the testing done by officers of the Mines Branch, Ottawa. It was the co-operation of these gentlemen that made the investigation possible, irrespective of any claims as to the merits of the investigation itself.

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Operations of Western Dominion Coal Mines, Limited

By K. JOHN *

(*Annual Western Meeting, Calgary, Alta., October,* 1951)

(*Transactions, Volume LV,* 1952, pp. 296-298)

Introduction

THE most recent estimate of mineable coal reserves in Saskatchewan was made by the Federal Royal Commission on Coal (1944-6). Saskatchewan was then estimated to have some 24 per cent of the Canadian reserves and two-thirds of this is located in what is called the Souris-Estevan area, which lies in the southeast corner of the Province and borders on the International boundary. It is also estimated that there are some 6½ million thousand-tons in this area of lignite coal which grades 8,000 B.t.u. or less and has a moisture content ranging from 30 to 33 per cent.

Ownership of coal rights in the Province is widely dispersed between the Crown, individuals, and corporations, and a recent survey disclosed that 40 per cent of ownership was vested in the Crown and 60 per cent in private individuals and corporations.

Coal rights on Crown lands may be leased at an annual rental of $1.00 an acre. Under such leases, a royalty of 5 cents a ton is levied on all coal mined. A royalty of 7 cents a ton is levied on coal produced from lands where mineral rights were purchased from the Federal Government prior to the transfer of jurisdiction over natural resources to the Province, in 1931. An annual mineral tax of 50 cents per acre has also been imposed by the Provincial Government on all coals within a defined producing area.

According to official records, coal mining as an industry commenced in Saskatchewan in 1887, when 400 tons of lignite was produced. These early operations were confined to the banks of the Souris river, near the town of Estevan, where outcropping seams could be readily mined. The coal was floated down Souris river to Winnipeg. Returns show that, in 1950, over two million tons was mined.

Increasing production has been characterized by a marked decrease in the number of operating units and by an equally important concentration of production in the Estevan field. Thus, whereas there were some 122 mines in Saskatchewan in 1940, this number has been reduced to about 60. The Estevan field accounts for about 99 per cent of the total Provincial output and some 95 per cent of the production is by four companies, Western Dominion Coal Mines, Limited, alone accounting for over 50 per cent.

There are several seams of lignite in the Estevan field. The two main seams are those known as the *Lower Seam,* 6 to 10 feet thick with an overburden of approximately 120 feet, and the *Upper Seam,* 5 to 10 feet thick with an overburden of from 15 to 50 feet.

Prior to 1930, the Lower seam was mined by the room-and-pillar method but, with the introduction in that year of strip methods, the larger producers gradually turned to the Upper seam. Today, practically the whole of the production from the Estevan field is obtained by strip mining, in the adoption of which practice Western Dominion was the pioneer.

Company Holdings and Reserves

Western Dominion Collieries, Limited, started operations at the turn of the century, when they opened a shaft mine at Estevan, and since that time they have been the largest operators in the field. In 1930, the Truax-Traer Coal Company, of Chicago, commenced a stripping operation near Estevan and in 1939 they amalgamated with Western Dominion and moved to the present site at Taylorton, four miles south of Bienfait and nine miles east of Estevan. At that time, the Company was re-organized as a subsidiary of the Great West Coal Company, Limited, with Truax-Traer reponsible for the operation and Great West Coal for sales. In 1944, Truax-Traer sold their interest to the Great West Coal Company and the whole operation is presently carried on by Western Dominion Coal Mines, Limited, wholly owned subsidiary of Great West Coal.

In passing, it might be mentioned that, during the war emergency, the Company re-opened its shaft mine at the request of the Dominion Government, in an effort to get additional tonnage. This work, however, was discontinued at the end of the war.

The Company owns some twelve thousand acres of coal lands and it is estimated that, in the Upper seam, they have approximately 38 million tons which can be economically mined with present equipment and, in the Lower seam, an additional 40 million tons, the mining of which might require larger equipment. Over most of its area, the property has been tested to a depth of 50 feet by home-made rotary drills and to further depths by larger drills.

Stripping and Cleaning Seam

The first operation in strip mining is, of course, the removal of the overburden. This is presently done by three power shovels:

(1) A *320-B* Bucyrus-Erie crawler-type electric shovel, powered with a 450 h.p., 4,000 volt, a.c. synchronous motor, driving d.c. generators for hoist, swing, and thrust motions. This shovel weighs approximately 520 tons and has a 90-foot boom, a 57-foot dipper handle, and carries a 10-yard dipper. It makes an average operating cycle of 45 seconds or, in other words, moves approximately 500 cubic yards of earth per hour.

(2) A *5W* Bucyrus-Monighan Walker-type electric dragline, powered with a 200 h.p., 4000 volt, a.c.

*Secretary-Treasurer, Western Dominion Coal Mines, Limited.

Figure 1.—500W Ward Leonard Bucyrus-Erie electric walking dragline, 165-foot boom, 15 cu. yd. bucket.

induction motor, driving d.c. generators for hoist, drag, and swing motions. This dragline weighs approximately 200 tons and has a 105-foot boom with a 5-cubic-yard bucket. Its operating cycle is 35 seconds, moving approximately 400 cubic yards per hour.

(3) A *500W* Ward-Leonard Bucyrus-Erie electric walking dragline, powered with two 450 h.p., 4,000 volt, a.c. synchronous motors, driving d.c. generators for hoist, drag, and swing motions (Figure 1). This dragline weighs approximately 720 tons, has a 165-foot boom, and carries a 15-cubic-yard bucket. Its operating cycle is 30 seconds, moving approximately 900 cubic yards per hour.

Up to the present, stripping has been continued the year round, even though cost increases, due to heavy frost conditions in the winter, are entailed. But since the inception of the larger dragline, it is expected sufficient tonnage can be uncovered prior to the depth of winter. To contend with frost conditions, blasting is necessary, which is done by drilling into the side-wall as well as from the top. In some cases success has been obtained by spreading and setting fire to stove-size coal ahead of the dragline or shovel. This has removed the top frost sufficiently for easier operation.

It might be mentioned here that some success was obtained, in certain favourable areas, by the use of a 'Bladeveyor', made by the Euclid Company, of Cleveland, Ohio, which is pulled by a *D*8 caterpillar and which elevates the overburden into wagons (coal haulers with dirt boxes in place of coal boxes) to be hauled away. This same machine can load out the coal seam in like manner. Since acquisition of the larger dragline, this method of overburden removal has been discontinued.

After the overburden has been removed, the surface of the coal seam is cleaned, using bulldozers and power sweepers, prior to being blasted ready for the coal-loading shovels (Figure 2).

Loading and Haulage

Four loading shovels are in use, as follows:

Two *50-B* Bucyrus-Erie electric shovels, carrying specially designed 3¾-yard coal dippers.

One *420-B* Bucyrus-Erie electric shovel, carrying a 2¾-cubic-yard specially designed coal dipper.

One 7-yd. electric shovel which was built in the Company's own workshop (Figure 3).

At present, the haulage equipment consists of seven Euclid Diesel 20-ton capacity coal haulers and three of 40-ton capacity. These transport the coal from the open pits to the screening plant, over some 20-odd miles of Company-owned roads.

Tipple

The coal is unloaded over a ramp into a hopper and carried up an apron conveyor to the main breaker and thence to the screening plant. This was originally built for a capacity of 300 tons per hour, but varying demands have necessitated many changes to the original tipple, both to increase its capacity and to provide the various sizes demanded, until presently its capacity is approximately 900 tons per hour and some 8 sizes are obtainable.

Primary screening is done with a 10 ft. by 50 ft. double deck shaker screen and the remainder of the

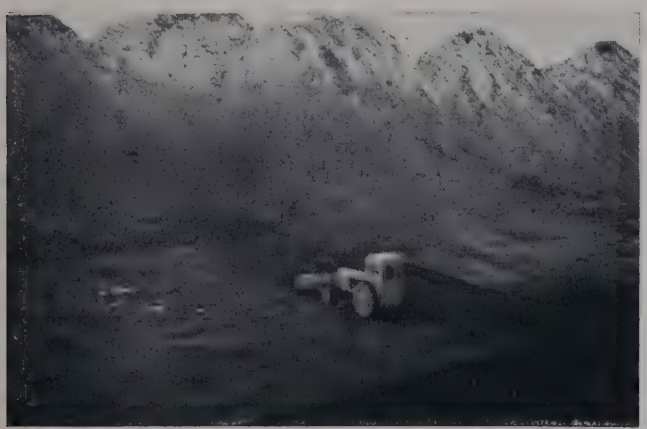

Figure 2.—Cleaning surface of coal seam.

Figure 3.—7-yd. electric shovel, built in Company's workshop.

screening with Dillon and vibrator screens, numbering 15.

At present there are two secondary crushers and a third is being installed. They are 30 in. by 60 in. double roll.

Western Dominion have been the leaders in the 'oil-treating' of coal, commencing as far back as 1940. This 'oil-treated' coal is sold under the name *Viking-ized*. The tipple is equipped for its production, providing a treated coal which is the final word in dust prevention and has a good appearance. Demand for this treated coal is steadily increasing.

Many sizes are, of course, loaded into open-type cars, requiring no loaders, but, for box-car loading, some eight loaders are installed, being the Wilson and Ottumwa type.

The tipple is operated by a main operator, who electrically controls the varied disposition of the coal to be screened. Under full production, some twenty-five men are needed in its operation.

To load some 7,000 tons, utilizing approximately 150 railway cars, in an 8-hour shift, requires a large complement of rolling stock, and, inasmuch as the annual output of 1,000,000 tons is mainly loaded in eight months, possible difficulties in this regard can be appreciated.

Shipping

Empty cars are fed into the tipple by a Diesel locomotive, and switching of loads is done by steam locomotives. The loaded cars are drawn by a steam switch-engine from the tipple to the Company scale-house, where they are scaled. After being marshalled in the Company yards, according to the railway to which they are consigned, they are drawn over a Company railway to the shipping point at Bienfait, some four miles away, and delivered to either C.P. or C.N. yards.

Railway equipment comprises three steam locomotives and one Diesel and all the necessary maintenance equipment such as roundhouse, locomotive shop, and track maintenance equipment.

Production

Production has steadily increased since 1939 and now has reached one million tons annually, with an overall per man-day tonnage of 25 tons, which, in a peak production month, rises to as much as 45 tons per man-day.

General

All records, accounting, etc., are kept by Western Dominion Coal Mines, Limited, at Taylorton (the plant site) but, after delivery of the coal to the railway, the Great West Coal Company (the parent Company) handles all invoicing, etc. The scale-house staff are responsible for the lining-up of the tipple, scaling of cars, billing, and the direction of railway crews.

A central heating plant provides heat to the offices, stores, garage, workshop, etc.; the 50-odd employees' residences are heated by space heaters or by furnaces.

The Company purchases electric power from the Saskatchewan Power Corporation, a Provincially-owned network. Electricity is supplied at reasonable rates, since coal for boilers at the Estevan plant is naturally plentiful. Supplementing the Provincial power supply, and standing ready at the mine, is an auxiliary plant that is used only in case of emergency.

Some hundred and fifty employees have all-year-round employment and, for their convenience, there are provided facilities for such recreations as baseball, skating, and various indoor games in an up-to-date Community Hall.

A central firehall is located in the camp, equipped with mobile fire engine.

An up-to-date workshop is maintained, in which all maintenance repairs are made. All servicing, repairs, etc., to haulage equipment are done at the plant.

For some years, provision of an adequate water supply for the plant was a problem, but this was eventually solved by the installation of a treating plant and by use of water obtained from the old underground workings.

A cafeteria is operated in conjunction with individual room accommodation for single employees, and education facilities are provided by a two-room school, operated by the Estevan School Unit.

The Company operates a general store for the convenience of employees, and a post office is located in the camp.

An up-to-date First-Aid room is maintained with a fully qualified man in charge and a motor ambulance ready at all times.

A funded-trust type of pension plan is in vogue, providing for a 5 per cent deduction from wages with a matching contribution by the Company, which, in addition, pays all administrative and carrying charges. A Group Life Insurance is available at a cost of 80 cents per month per $1,000 to all employees, regardless of age, the Company paying the balance. In addition, a Sickness and Accident "off the job" Insurance ($25 per week), for which the Company pays a monthly premium of 97 cents and the employee $1.11, is available. Workmen's Compensation rates are presently 7½ per cent of gross payroll.

Approximately one-third of the employees are resident at the mine, with the balance equally spread between Estevan and Bienfait.

Since the inception of the present Company in 1939, the men have been represented by the Saskatchewan Coal Miners' Union, affiliated with the National Council of Canadian Labour, and at no time have there been any labour difficulties, contracts being negotiated annually.

Western Dominion coal is sold under the trade name of *Klimax*.

Mr. J. R. Brodie, of Winnipeg, is President of the Company, with Messrs. F. H. Nord and R. W. Wilton, of Brandon, Directors. C. F. Doerr has been Mine Superintendent and Director since 1943.

Canada's Petroleum and Natural Gas Resources

*By G. S. HUME**

(*Ottawa General Meeting, January*, 1952)

(*Transactions, Volume LV*, 1952, pp. 299-301)

Introduction

THERE HAS BEEN a marked growth in the petroleum industry in Canada since the end of the war. The rate of growth in the demand for oil products has exceeded that in United States, although the *per capita* consumption is still less. Estimates of the consumption of petroleum and petroleum products in Canada in 1951 are approximately 154,000,000 barrels, which is slightly more than 11 barrels *per capita*, whereas the estimated consumption in United States for the same period, according to *World Petroleum*, is placed at 2,619,970,000 barrels. This, on the basis of a population of 154,000,000, would be equivalent to 17 barrels *per capita*. The increase in consumption of the petroleum and products is world wide. World production in 1951 has been estimated (*Oil and Gas Journal*) at 11,700,000 barrels a day or more than 4¼ billion barrels for the year. United States production increased 13.7 per cent to 6,143,000 barrels a day and Canada's total production increased about 65 per cent from a total of 29,091,235 barrels in 1950 to an estimated (*Dominion Bureau of Statistics*) 48,096,800 barrels in 1951.

The trend to use increased amounts of petroleum and petroleum products, particularly in the United States and Canada, has also been supplemented by increased use of natural gas. The rate of expansion of the consumption of natural gas in United States in the past few years has been very rapid, increasing from 2,600 billion cubic feet in 1940 to 5,975 billion in 1950, with the proven reserves in the same period increasing from 85,000 billion to 186,000 billion cubic feet. In this respect it is interesting to note that, even with the tremendous increase in the amount of proven reserves in the past few years in the United States, the rate of increase in consumption has been even faster, with the result that the ratio of reserves to production has been declining. At the end of 1945, the reserves were equal to 30.5 years at the then rate of production whereas there was 27.4 years' supply in relation to the proven reserves at the end of 1950. This fact was discussed and may have had some influence in the recent refusal of the Federal Power Commission to allow export of gas from Texas into southwestern Ontario. The figures for natural gas production in the United States for 1951 are not yet available but they will undoubtedly show a considerable increase over 1950. At the end of 1950, a new 1,900-mile pipe line was supplying natural gas from the Gulf Coast region to New York, and, in addition, San Francisco was linked by pipe line with the gas fields of West Texas, New Mexico, and the Texas Panhandle.

In considering Canada's fuel needs in relation to possible supplies, therefore, it is well to keep in mind pertinent facts in relation to developments in the United States which directly affect the economy of this country. The United States is now a net importing country for petroleum and it is reasonably certain that this will continue at an increasing rate even though domestic production has been and will be still further increased. The need for future supplies of liquid fuels in the United States is being considered in relation to long-term plans and the value of the Colorado oil shales is being determined by field operations. Also there is increasing activity in research to transform coal or heavy oils to light oils by hydrogenation, and research also is proceeding on transforming natural gas into liquid fuels. In fact one plant was built to try to do this commercially, but it has not been wholly a success.

All this points to the assumption that, in the United States, consumption of petroleum and petroleum products is expected to increase and a similar situation can, with equal assurance, be predicted for Canada. It is, therefore, timely to try to assess our resources of oil, natural gas, and bitumen in Canada in order to be able to form an estimate of our hydrocarbon fuels in relation to the much greater volume of fuel represented by our coal deposits in eastern and western Canada. It should always be kept in mind that coal will ultimately be the main source of our heat energy until such time as radioactive materials can be used to advantage, but for the present coal must compete with the more easily adaptable fuels such as oil and gas, as these become more readily available.

Petroleum Supplies in Canada

The facts regarding the expansion of exploration and discoveries of oil in western Canada are familiar to most Canadians. The discovery of the Leduc oil field in 1947, followed by the further discovery of Redwater and other fields in Alberta in 1948, has led to an exploration campaign that has resulted in light to medium oil discoveries in Saskatchewan, Manitoba, northeastern British Columbia, and the extension of exploration into the Northwest Territories. Crude oil has now become available in increasing amounts, not only leading to large-scale refinery expansion in the Prairie Provinces but also to the building of a pipe line outlet for crude oil to the head of the Great Lakes in the United States for delivery of oil mainly to Ontario refineries. These, in turn, have been enlarged or new ones built to keep pace with the supply from the new source.

*Director General of Scientific Services, Department of Mines and Technical Surveys, Ottawa.

Recently, the Board of Transport Commissioners has given permission to build the Trans-Mountain oil pipe line from Edmonton to the Pacific coast. This 24-inch line will initially handle only about 37,500 barrels a day but with the addition of three more pumping stations could handle 200,000 barrels. Three refineries in the Vancouver area will be enlarged to refine this crude oil, but in the immediate future none of it is likely to go to the United States on account of the high import duties. Thus, for the present, the refineries in the Vancouver area will be adapted to the possible market in the coast region of British Columbia. It is proposed that pipe line construction will start in 1952 and be completed in 1953. In the meantime, therefore, the market for Alberta crude oil will be limited to the Prairie Provinces and to oil handled to Ontario and the United States by the Interprovincial pipe line. In spite of this limitation of markets, the search for new oil fields has continued in Alberta at an unprecedented pace.

Recently, at the hearings of the Board of Transport Commissioners, an estimate of the petroleum reserves of western Canada was presented by A. I. Levorsen, Stanford Research Institute. This estimate is thus an authoritative opinion and in summary is as follows:

	Barrels
Estimated oil in place October 1, 1951	3,272,000,000
Net estimated recoverable oil (less Norman Wells, Peace River area, and Lloydminster area heavy oils) (40% recovery factor)	1,330,000,000
Estimated recoverable reserves as of December 31, 1953	1,886,000,000
Estimated ultimate reserve	Several times the present reserve.

According to Levorsen, "the average rate of production in United States over a number of years ranged between 7 per cent and 8 per cent of the recoverable reserves per annum". The Alberta Petroleum and Natural Gas Conservation Board has approved a current rate of permissible production at approximately 6 per cent per annum of the proved recoverable reserves. Thus Levorsen arrives at the conclusion that "the maximum efficient rate of annual production (MER) will eventually be around 6 per cent to 8 per cent of the recoverable reserves". The figures he gives of possible daily production are shown in Table I.

Table I.— Possible Daily Oil Production, Western Canada, at Efficiency Rates Noted

	6%	7%	8%
MER, Oct., 1951	218,000 b/d	255,000 b/d	291,000 b/d
Est. MER, 1954	307,000 b/d	358,000 b/d	409,000 b/d
Est. MER, 1955	336,000 b/d	390,000 b/d	453,000 b/d

In 1951, according to the *Nickle Daily Oil Bulletin*, the production in the Prairie Provinces was 47,725,000 barrels, or more than 10 million barrels greater than in 1950. This means an average production of 130,750 bbl. a day for 1951, or approximately the present production rate during the winter months. It is thus obvious that the supply exceeds the market demand, and with the expected discoveries, based on the past record, it is anticipated there will still be a considerable excess potential production when the Trans-Mountain pipe line is built.

The enlargement for markets might reasonably be expected to be in the United States. At present, the tariff on crude oil entering the United States is at the rate of 10½ cents a barrel on a small quota and, when this is filled, the tariff is 21 cents a barrel. Thus, although crude oil is now available in Canada, it is by no means certain to what extent it will be marketed in the United States.

For the present, therefore, there is an excess crude oil supply in western Canada, even though domestic production is only about 33 per cent of our consumption. It is likely that a larger part of our requirements in the next few years will be filled by our own production. However, since about 40 per cent of the refining capacity of Canada is in the Montreal area, it is questionable how much oil from western Canada could be delivered to this area at a competitive economic rate even when the St. Lawrence seaway is in operation. It is a safe inference, therefore, that, because of transportation costs, Canada will never entirely use Canadian-produced oil for all its refineries, but it is highly probable that, on balance, we may become a net exporting country with exports into foreign markets considerably exceeding imports from abroad into eastern Canada.

Natural Gas Supplies in Canada

As already indicated, there has been a tremendous increase in the use of natural gas in the United States, and the proven reserves have grown enormously with the demand. With this growth the public, both in the United States and Canada, has become conscious of the value of natural gas as a fuel and interest in the gas resources of western Canada has been aroused by the possibilities of export of gas from Alberta to other parts of Canada or to the United States.

In Alberta, the Petroleum and Natural Gas Conservation Board has held hearings concerning the natural gas situation, in which evidence has been presented by companies seeking export permits. In an interim report of the Board, issued at the end of 1950, it was concluded that the established reserves of disposable gas amounted to 4,658 MMM.c.f. and that the requirements for the Province of Alberta for the 30-year period 1951 to 1980 were 3,059.9 MMM.c.f., with an established reserve of 4,500 MMM.c.f. required to meet this need.

Estimates of natural gas reserves have been made by various geologists and engineers. Two reports and a supplement to the first report were issued by the Department of Mines and Technical Surveys. In the first of these, the natural gas *in situ* at 100 p.s.i.a. and 60°F. as of November 30th, 1948, was placed at 4,207 MMM.c.f. for Alberta and 65 MMM.c.f. for Saskatchewan, or a total of 4,272 MMM.c.f. A further report as of June 30th, 1950—that is, 7 months later — placed the proven and probable reserves *in situ* at the same pressure and temperature at 6,400 MMM.c.f., or an approximate increase of 2,100 MMM.c.f. Thus the rate of growth as new discoveries were made or new extensions proven is indicated by these figures. In 1951, evidence presented by the engineering firm of DeGolyer and MacNaughton, Dallas, Texas, on behalf of Trans-Canada Pipe Lines, Limited, gave the recoverable reserves for Alberta as of August 1st, 1951, as shown in Table II.

These figures indicate a very considerable surplus over Alberta's needs for the next thirty years.

TABLE II.— ESTIMATED RECOVERABLE RESERVES, ALBERTA, AUGUST, 1951

	MMc.f. AT 14.4 p.s.i.a. AND 60°F.
Proved	8,553,316
Probable	2,509,517
TOTAL, Proved & Probable	11,062,833
Possible	2,644,623
TOTAL	13,707,456
Available for sale	8,415,442
Deferred by oil production	460,773

It is quite apparent that, until a market is available, the drilling for natural gas alone in Alberta has no immediate economic incentive beyond the hope of proving-up further reserves in the expectation that, when sufficient are indicated, export permits will be given. Thus it is confidently expected that, if and when one or more export permits are given for long distance transmission lines to carry gas, the reserves will be increased at a much greater rate than at present, even though the present rate of increase appears large. Such a trend can be predicted with confidence in view of the history of development of gas reserves in the United States.

The possibilities of utilization of gas from different parts of Alberta by the building of pipe lines to other parts of Canada or to the United States have already been discussed (1). In the hearings in Alberta during the latter part of 1951, the needs of Ontario were presented by its Fuel Commissioner following the denial by the Federal Power Commission of the United States to allow export of Texas gas *via* Detroit into southwestern Ontario, where natural gas has been used more or less since 1890 to the extent of the available supply, which is now relatively small in relation to the demand. Southern Ontario and southwest Quebec, in addition to all centres of population *en route* from Alberta across the Prairie provinces and northern Ontario, are the largest prospective outlets for natural gas in Canada and, except on the Prairies, these areas are deficient in fuel supplies. The advantages of a supply of natural gas for them from Canadian sources are obvious, and especially would such a supply prove a boon to northern mining communities.

The situation in respect to the Pacific coast is somewhat more complicated in that the evidence seems to indicate it would not be economical to build a pipe line from Alberta to supply the Vancouver area alone, since the market is too small. However, the Pacific Northwest area of the United States offers a large market not yet supplied by any natural-gas pipe lines and is much nearer to the source of supply than is the market in Ontario and Quebec. It is, therefore, a more economic market, a factor which will doubtless be properly considered if and when a decision to export gas from Alberta is made. From the standpoint of British Columbia, however, the development of the gas and oil reserves in the northeastern part of the Province is of primary concern, and considerable quantities of gas are likely to be obtained in the production of oil, the first commercial discovery of which was made recently in the Fort St. John area. Gas — for which there is no local market — when it accompanies oil production, presents a problem of conservation, and an outlet for it is highly advantageous in order not to retard oil exploration and development. Such an outlet could be provided by a gas pipe-line joining the Peace River area either with the Vancouver area or with the Edmonton area, from which other outlets might be made available. Undoubtedly, all the advantages will be given proper consideration when decisions regarding pipe line routes are made.

All this points to the fact that western Canada has now reached the stage of development where enormous oil and gas reserves are indicated. The effective use of the gas will have a profound influence on the economy of this country not only in stimulating chemical and other industries such as have recently been located in Alberta but in supplying fuel to those parts of Canada within economic reach by the relatively cheap pipe line method of transportation.

(1) HUME. G. S., *Utilization of Natural Gas in Canada;* Eng. Jour., Vol. 34, Sept., 1951, pp. 853-860.

THE BITUMINOUS SANDS OF THE MCMURRAY AREA

No long-term policy of energy development can avoid taking note of the bituminous sands of northern Alberta. A symposium sponsored by the Alberta Government and held in Edmonton last September, following the publication of an exhaustive report by S. M. Blair on the commercial feasibility of development, has again focused attention on one of Canada's richest undeveloped resources. The time seems to be close when plans will be laid for large-scale operations, and the evidence being gradually accumulated by scientific investigations points to economic possibilities in the very near future. Indeed, to some the time seems to have already arrived to plan for future production. The limitations of the market for oil from western Canada in the immediate future need not be a prime concern since it is obvious to all familiar with this problem that some years must elapse before any plant would be in a position to produce significant supplies from the bituminous sands and there can be no doubt whatever about the ultimate continental requirements. However, a very large amount of risk capital is needed, since it is generally agreed that a plant to be economically successful and to justify economic transportation of oil by pipe line to connect with marketing facilities must be of the order of 100,000 barrels a day. It may be that from the strategic standpoint, in order to secure continental supplies for an emergency, some agreement with the United States would be of great mutual benefit in obtaining development of the bituminous sands by ensuring an outlet when a plant is brought into production. In view of the expected surplus of crude oil from wells in western Canada, a domestic market for bituminous-sand oil does not seem likely at least in the next decade or two, except in the event of some unpredictable demand such as might speedily be brought on by a war crisis. Thus, although the reserves of bituminous-sand oil are large and methods of extraction that have economic possibilities are known, the immediate development of the sands is by no means certain.

A Report of the Research Committee Coal Division C. I. M.

Progress in Coal Cleaning

With Special Reference to the Fines Problem

By E. SWARTZMAN *

(*Ottawa General Meeting, January,* 1952)

(*Transactions, Volume LV,* 1952, pp. 302-307)

Introduction

DURING 1951, experimental cleaning studies on both fines and larger sizes of certain Canadian coals were conducted on a relatively large scale in two pilot plants in the United States, large enough to be considered as small commercial units, and in one commercial plant in Canada.

One of the pilot plants was a wet washer of a type which apparently has never been used in Canada, nor has any Canadian coal been tested in it before. In the other pilot plant and in the commercial plant, dry cleaning equipment was used. The commercial plant is the first of its kind to be employed in Canada, and this type is used only to a limited extent in the United States.

Fine-Coal Cleaning Experiments

Test Plants

The wet washer employed was a Hydrotator and Hydrotator-Classifier combination pilot-plant with a capacity of about twenty tons per hour. This is a classifier type washer adapted for cleaning fine coal as well as certain intermediate sizes. The Hydrotator deals with that portion of a coal retained on approximately a 24-mesh Tyler sieve, whereas the Hydrotator-Classifier deals with the minus 24-mesh fines.

Basically, the Hydrotator (Figure 1) and Hydrotator-Classifier (Figure 2) consist of a system of hollow arms radiating from a central distributing manifold, each arm having several downward inclined nozzles. When water is discharged from these inclined nozzles the impulses derived from the resultant jet rotates the agitator in a horizontal plane within a cyclindrical tank. Water mixed with undersize material, from the clean coal dewatering screen in the case of the Hydrotator unit and from within the tank itself in the classifier unit, is pumped back through the agitator, creating upward currents which float out the clean coal. The fine solids in circulation build-up the medium in the tank to a gravity which is determined by the characteristics of the coal being cleaned and an automatic float control. This float control, which admits clean water into the system, utilizes the gravity differential between the main classifier tank and the underflow conveyor to control the discharge of refuse.

In order to facilitate the cleaning of the finer sizes (approx. minus 24-mesh material), the classifier unit, in addition to having a tank with a much greater diameter, has certain other modifications in comparison to the Hydrotator unit. The circulating pump, instead of being external as in the Hydrotator unit, is submerged in the tank and is located in the central supporting column of the agitator. The circulation thus is confined to a zone between the tank bottom and the pump intake level. The feed is introduced at one side of the tank instead of in the centre, and is withdrawn at practically the same level at a point farthest distant from the feed. This allows much more time for the refuse to settle as the coal travels horizontally in suspension. In addition, to effect further separation of entrained coal particles, the refuse settles through a second stage at the bottom of the tank, where it meets a controlled upward current of clear water introduced through the second stage agitator, which is firmly attached to the main agitator and turned by it.

The pilot plant scale Airflow dry cleaner (Figure 4) with a capacity of about five tons per hour and designed specially to handle finer sizes of coal, but adaptable also for coarser sizes, consists essentially of a reciprocating perforated deck mounted over an air box. The coal moves forward under the influence of forces created by the longitudinal downward slope of the deck, pulsating air currents, the oscillating action of the deck, and the crowding of incoming particles. Stratification is somewhat similar to that effected on the American Table, the lower layer next to the deck surface being composed mainly of coarse coal, on top of which rides the fine coal. Within the machine, the resistance pack below the perforated deck surface is divided into a number of cells containing resistance material, such as marbles. Below this, a number of air valves made of slotted plates which slide over each other control the distribution of air, a rotating shutter in the inlet providing a pulsating air current. In the deck there are several wells extending across the full width for refuse discharge, small screw conveyors receiving the refuse from each well and discharging it at the side of the machine. That refuse coming off near the discharge or clean coal end of the table is usually a middling product and is normally re-circulated for re-cleaning.

Both of the above types of plant are marketed by the Roberts and

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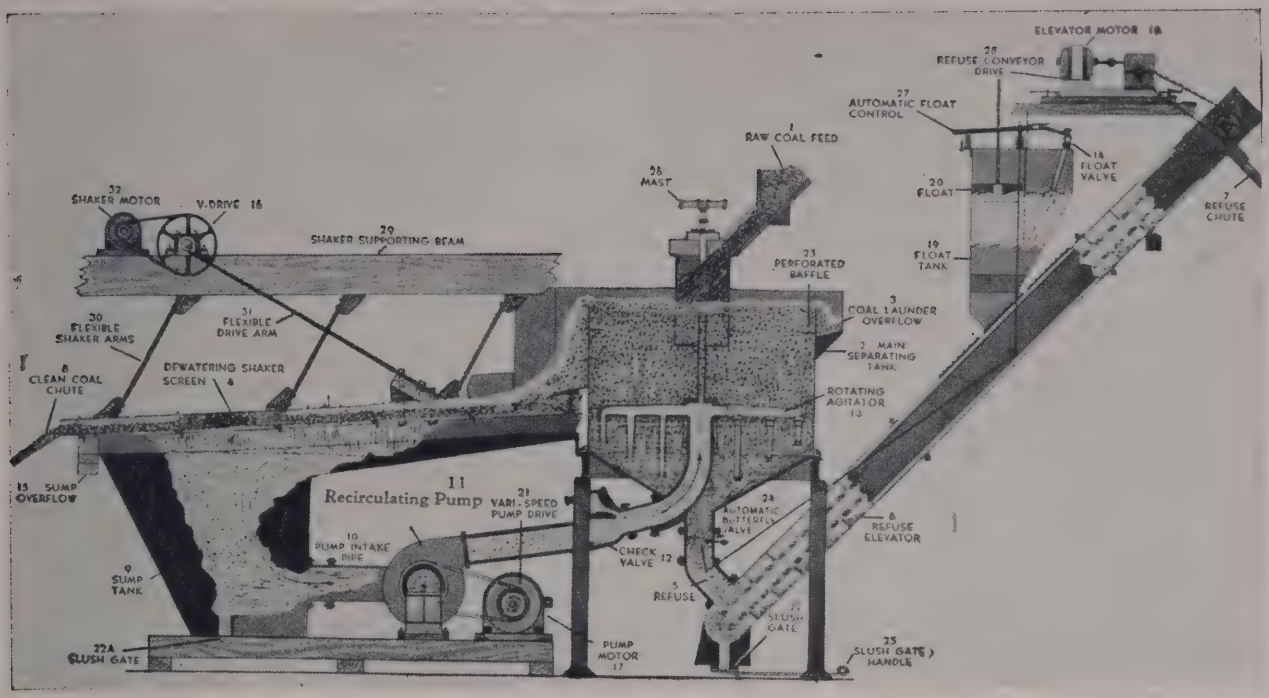

Figure 1,—Hydrotator process unit.

Schaeffer Company, Chicago, Ill., and the experimental pilot plants are situated on their factory site at Harvey, Ill., where the experimental cleaning tests (on two Western Canadian coals) were conducted.

Test Results

Single carload samples of approximately 50 tons of each of two Western Canadian coals, namely, 0 x ¼ in. medium volatile coal from the Crowsnest area of Alberta and 0 x ½ in. similar rank coal from the Mountain Park area of Alberta, were tested to determine their amenability to cleaning by means of the two processes.

The raw Crowsnest area coal contained 22.5 per cent ash whereas the raw Mountain Park area coal contained 17.4 per cent, and neither coal, according to float-and-sink data, is an easy coal to clean because of their high inherent ash and because of the relatively large amount of boney, intermediate gravity, middlings material present. The float-and-sink data indicated that, using 10 per cent on the ±0.10 specific gravity distribution curve as the gravity at which a separa-

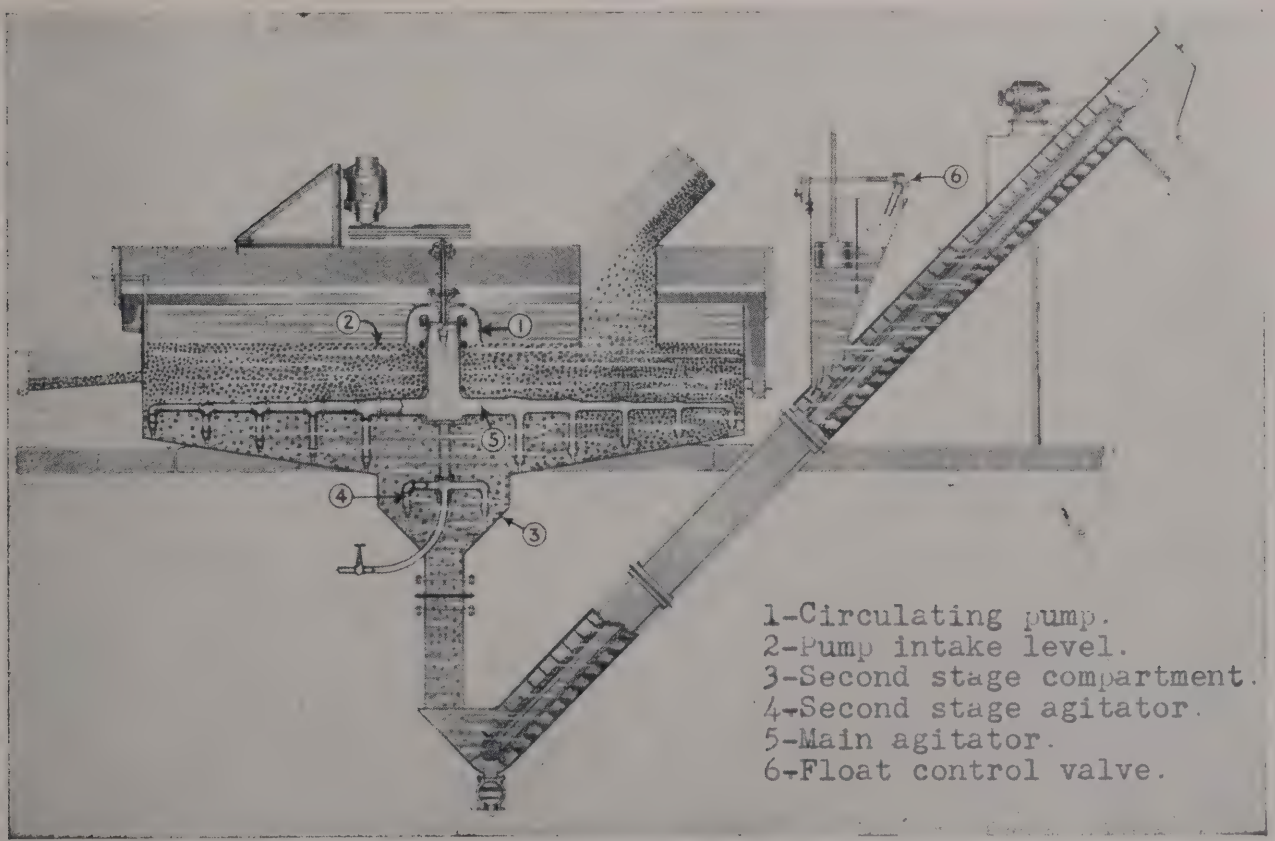

Figure 2,—Hydrotator classifier unit.

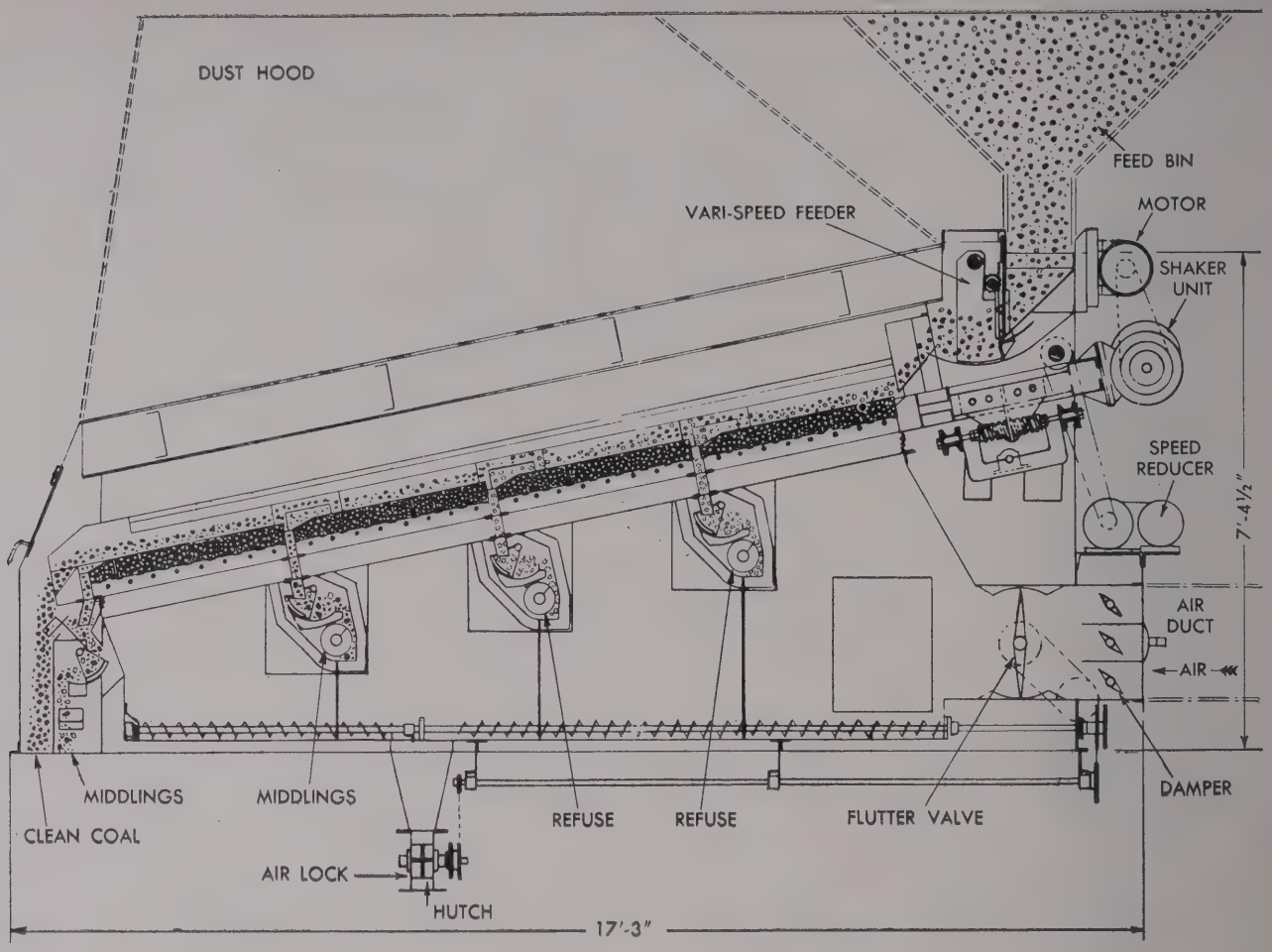

Figure 4.—Airflow pneumatic cleaner.

tion could be readily and efficiently made in most wet washers, the coals could be cleaned as follows:

	Crowsnest Coal	Mt'n Park Coal
Separating G....	1.63	1.57
Yield, clean coal.	79.0%	82.0%
Ash, clean coal...	11.5%	10.4%

Cleaning in a pneumatic machine is considered generally to be less efficient than in a wet washer and thus, for the same ash contents, lower yields should be expected, or, for the same yields, higher ash contents should be anticipated.

With the Hydrotator combination, the results shown in Table I were obtained.

Table I.—Results of Cleannig with Hydrotator Combination

The density distribution curves for the clean coal and refuse products of these coals indicated that the Hydrotator, with the plus 24-mesh material, was much more efficient than the Hydrotator-Classifier unit dealing with the minus 24-mesh fines. For example, in the case of the Crowsnest area coal, the comparison was as follows:

	Hydro-tator	Classi-fier
Effective separating G.	1.78	1.50
Refuse in clean coal..	2.8%	9.7%
Clean coal in refuse...	2.7%	9.3%

Thus there was more incorrectly classified material in the case of the Hydrotator-Classifier separation of the minus 24-mesh material than in the case of the Hydrotator separation of the larger sizes. In this regard it should be noted that the ash contents of the clean coals from the classifier separation were substantially lower (12.9 per cent for the Crowsnest coal and 7.7 per cent for the Mountain Park Area coal) than were those from the Hydrotator separation. However, to effect this there was a sacrifice in the yield, and good, clean coal was discarded with the refuse.

Elimination of the dust (mainly minus 48-mesh material) from the coal appeared, in the case of the Crowsnest coal tested in this manner, to improve the overall washing characteristics. The total yield of clean coal increased from 71.7 per cent to 88.9 per cent and the ash content was reduced from 13.5 per cent to 12.6 per cent. Specifically, the yield of coarser clean coal in the Hydrotator was substantially increased, with little change in ash content, whereas, in the case of the finer sizes separated in the classifier, the yield of clean coal was only slightly increased but the ash content was substantially reduced.

Table I.— Results of Cleaning With Hydratator Combination

	Crowsnest Area Coal		Mountain Park Area Coal	
	Yield %	Ash %	Yield %	Ash %
Total clean coal.....	71.7	13.5	81.8	11.8
Total refuse.....	16.7	53.6	10.0	57.0
Sludge.....	11.6	24.9	8.2	20.7
Raw coal.....	100.0	22.5	100.0	17.4

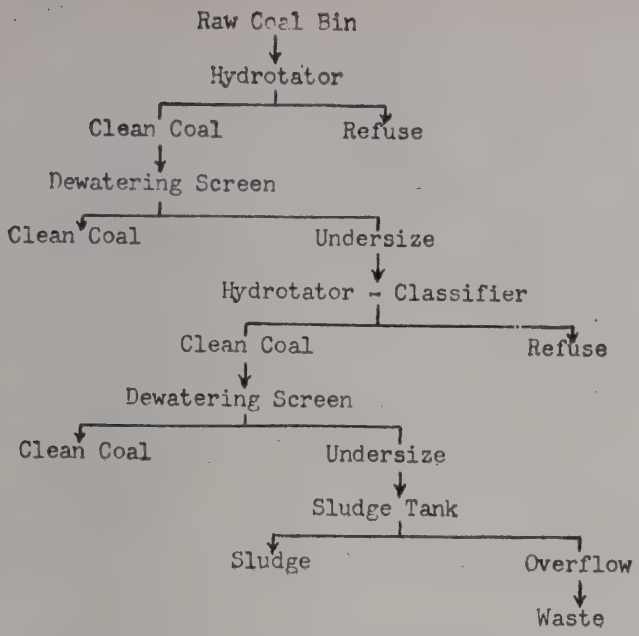

Figure 3.—Hydrotator and hydrotator classifier flow-sheet.

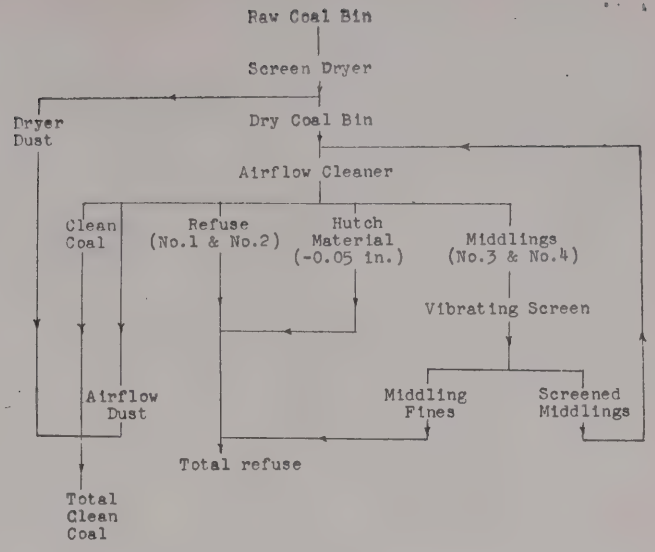

Figure 5.—Airflow flow-sheet.

The removal of the dust so improved the separation in the classifier that the incorrectly classified material amounted to only 10.7 per cent of the cost instead of 19.0 per cent. The improvement noted may have been mainly due to the improved viscosity conditions as a result of the removal of the fine particles of coal, and especially of shale.

With the Airflow dry cleaner, the results given in Table II were obtained.

The density distribution curves for the clean coal and refuse indicated that the separations in the air cleaner were not as efficient as those in the Hydrotator. In the case of the Crowsnest coal, for example, the effective separating gravity was 1.66, with 12.5 per cent of incorrectly classified material split evenly between the clean coal and refuse.

In all cases with both types of machine, the clean coal showed an increase in ash with a decrease in size. However, in the Airflow machine cleaning was not effective on material passing the 20-mesh screen, whereas, in the case of the Hydrotator combination, cleaning was effected on sizes down to the 48-mesh screen. It should also be noted that in the case where the dust was removed prior to processing in the Hydrotator combination washer, the intermediate fine sizes washed in the classifier section showed lower ash contents, and even such minus 48-material as remained in the de-dusted product exhibited substantial ash reduction.

Generally, the results of the above experimental pilot plant tests showed that the fines and smalls from both of the Alberta bituminous coals examined were amenable to cleaning by the wet Hydrotator process and the pneumatic Airflow process. Although efficiency of separation was better in the Hydrotator, when an ash content of about 13.5 per cent for the Mountain Park area coal and 14.5 per cent for the Crowsnest area coal is considered satisfactory, the dry cleaning process has certain obvious advantages in view of the fact that the clean coal as well as the dust is reclaimed in the dry state. If, however, lower ash contents are required, it is obvious that the wet cleaning method would be more suitable. In either case, a drying plant would have to be included, because normally the slacks as mined are usually too damp for satisfactory treatment in a pneumatic cleaning plant.

Coarse-Coal Cleaning Experiments

Test Plant

The commercial dry cleaning plant in which recent tests on a 5/16 in. by 4½ in. size coal were conducted uses the Air-Sand Process developed by Thomas Fraser and marketed by the Stephens-Adamson Mfg. Company of Aurora, Ill. (Figure 6). This commercial plant, installed at a mine in the Sydney Mines area of Nova Scotia, is the only known dry cleaning process utilizing a flowing dense medium. It is formed by bubbling air through a mass of dry sand, 24 x 65 mesh in size. Theoretically, the air fluidizes the sand mass, causing it to behave somewhat as a heavy liquid, the coal floating on the aerated sand and the refuse sinking. According to Fraser, by adjusting the rate of air flow, the specific gravity of the aerated sand medium may be varied between 1.4 to 1.6, the effective gravity of separation being a little higher than the static density of the medium due to the effects of viscosity and the carrying power of the moving currents.

The air-sand cleaner installed at the Nova Scotia mine, designed for coal from about 4 or 5 in. down to ¼ in., with a capacity for this size range of approximately 80 tons per hour, is the so-called standard four-foot unit and consists of a two-compartment inclined trough split longitudinally into a primary wash box 9 ft. long by 4 ft. wide and a parallel secondary wash box

Table II.—Results of Cleaning With Airflow Dry Cleaner

	Crowsnest Area Coal		Mountain Park Area Coal	
	Yield %	Ash %	Yield %	Ash %
Total clean coal	72.4	14.9	82.2	13.4
Total refuse	17.4	53.2	11.7	43.3
Total dust	10.2	18.5	6.1	18.0
Raw coal	100.0	22.5	100.0	17.4

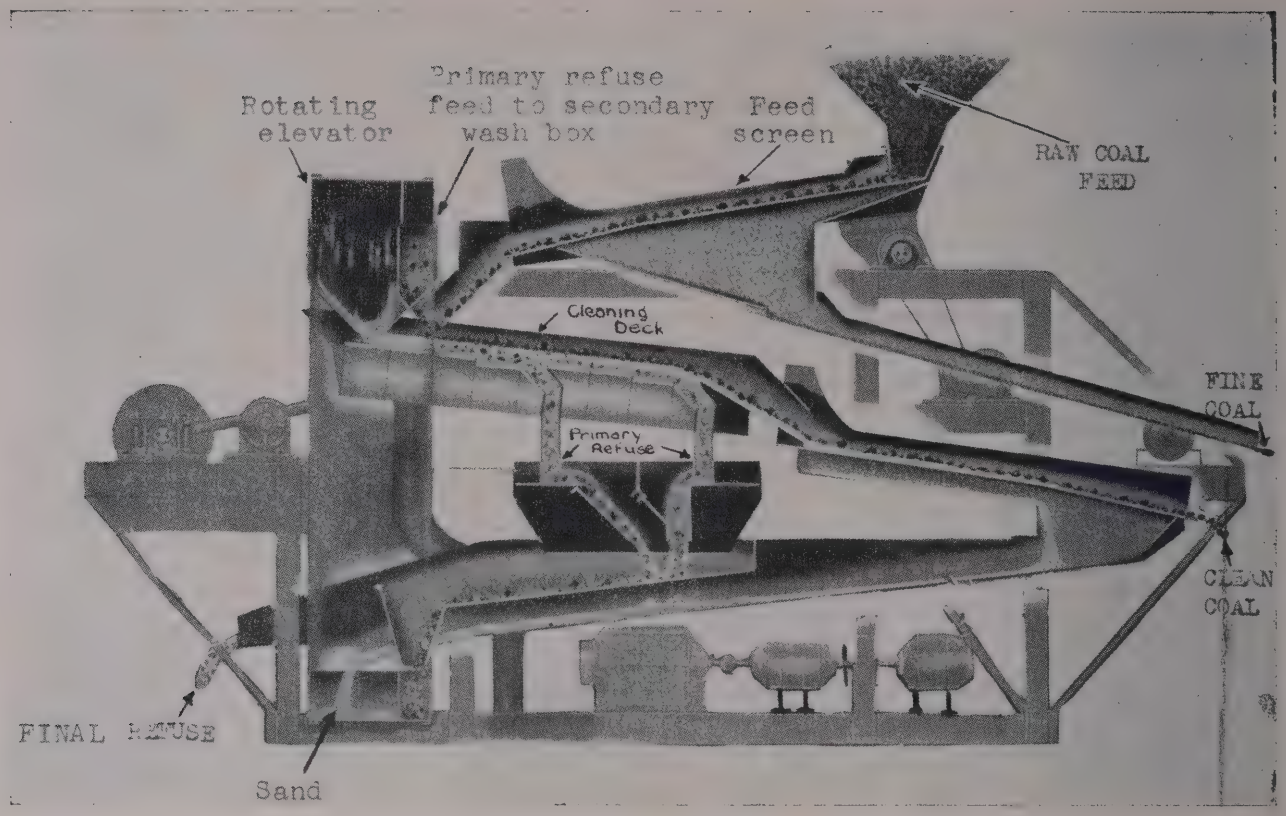

Figure 6.—Stephens-Adamson air-sand dry coal cleaner.

9 ft. long by 18 in. wide, each compartment being 3 feet long and separated from its neighbour by a refuse well. The floor of the table is composed of permeable carborundum blocks. The primary and secondary sections of the table are provided with their own air boxes, the air flow to each being supplied by separate positive-pressure blowers and independent regulation being effected by means of several valves in each compartment.

The raw coal, after screening to the desired size range, is delivered to the head end of the primary wash box, where it is mixed with sand, delivered by means of a rotary elevator, and exposed to the fluidizing action of the air. The clean coal passes over the end of the table and is freed of sand by screening, the sand being returned to the table by means of the rotary elevator which is so designed that it serves also as a conveyor-elevator for the intermediate refuse. The refuse products from the primary box are conveyed to the intermediate refuse compartments of the rotary elevator and delivered to the head end of the secondary wash box. The final refuse products from this wash box are discarded after de-sanding by screening.

A small pneumatic table of the American coal cleaning type, fitted with a carborundum permeable deck, is available for freeing a portion of the sand continuously from the fine coal and very fine sand produced by attrition. Fresh sand is added at the rate of about 1¼ pounds per ton of coal processed, there being about 2½ tons of sand per ton of coal in circulation.

Test Results

The coal tested was a high-volatile *A* bituminous coal from the Sydney Mines area, Nova Scotia. During the test period of one shift, some 140 tons of the 5/16 in. by 4½ in. size was tested. This coal, containing 17.1 per cent ash and 5.6 per cent sulphur, did not have a low inherent ash content, and the material sinking in 1.80 was rather low in ash at 60 per cent. In addition, as the size increased, the quantity of intermediate gravity material increased, indicating possibly greater difficulty in cleaning with an increase in size. Relatively efficient wet cleaning, according to the ±0.10 specific gravity distribution curves, should be expected with this coal at a separating gravity of 1.665, with a 79.0 per cent yield of clean coal containing 9.8 per cent ash and a refuse containing about 54 per cent ash. The actual plant test in the dry air-sand machine gave a yield of 80.3 per cent clean coal containing 9.3 per cent ash and 4.4 per cent sulphur, with refuse containing 51.5 per cent ash.

The bulk of the heavy refuse (material sinking in 1.80) was removed and only a small proportion of the middling fraction remained in the clean coal. The refuse contained only 8.1 per cent of material with 13.5 per cent ash floating at a gravity of 1.50. The density distribution data for the clean coal and refuse showed that cleaning effected a separation at an equivalent gravity of 1.612, with 3.0 per cent refuse products remaining in the refuse. Although this quantity of incorrectly classified particles may appear somewhat high when thinking in terms of a true gravity separation, considering the large quantity of intermediate material present in the raw coal it is doubtful whether any wet cleaning process, with the exception of the heavy-media type, could effect any better separation. All sizes from 4 in. by 5 in. down to ¼ in. by ½ in. were cleaned down to the same ash content, with yields varying in accordance with the ash content of the size fraction.

The process thus was found to perform in a manner approaching a true heavy-media process. If the sand is kept relatively free from low gravity coal fines and very fine sand, very little difficulty

should be experienced in regulating the apparent density of the air-sand medium. Because of the necessity to keep the sand dry in order to maintain fluidity, it is very important that the raw coal feed be fairly dry.

Concluding Remarks

The results of the experimental tests as summarized in the accompanying tabulation, indicate the possible application of different types of wet and dry cleaning processes to the varying conditions met in the Canadian coal industry, and show the necessity of further experimental work on the pilot and the commercial plant scale.

Special attention has been, and is being, given to the question of beneficiating fines, especially since problems concerning these products are bound to increase with mechanization and the trend toward smaller sizes. With this in view, the Division of Fuels is equipping its Fuel Research Laboratories at Ottawa with experimental and small pilot scale equipment of diverse, but conventional, type with which extensive investigations can be undertaken to determine the cleaning characteristics of fines from various Canadian sources. Such studies may result in the necessity to modify existing types of equipment and processes and encourage the development of new processes.

Table III.— Summarized Results

Wet Washing Test on Finer Sizes

Equipment Used: Combination of Hydrotator* (for +24 mesh coal) and Hydrotator-Classifier (for −24 mesh coal)

Capacity: 20 tons per hr.

Coals Tested: (Med. volatile) 0 x ¼ in. size from Crowsnest area, Alta., and 0 x ½ in. size from Mountain Park area, Alta.

	Crowsnest Coal	Mountain Park Coal
Raw coal	22.5% ash	17.4% ash
Clean coal	71.7% with 13.5% "	81.8% with 11.8% "
Sludge	11.6% with 24.0% "	8.2% with 20.7% "
Refuse	16.7% with 50.6% "	10.0% with 57.0% "

Dry Cleaning Test on Finer Sizes

Equipment Used: Airflow Dry Cleaner*, capacity 5 tons per hr. (adaptable for both fines and coarser sizes).

Coals Tested: Same as above.

	Crowsnest Coal	Mountain Park Coal
Raw coal	22.5% ash	17.4% ash
Clean coal	72.4% with 14.9% "	82.2% with 13.4% "
Dust	10.2% with 18.5% "	6.1% with 18.0% "
Refuse	17.4% with 53.2% "	11.7% with 43.3% "

Dry Cleaning Test on Coarser Sizes

Equipment Used: Air-sand cleaner† (adaptable for ¼ in. to 4 or 5 in. coal)

Coal Tested: 5/16 to 4½ in. coal from Sydney Mines area, N.S.

Raw coal	17.1% ash	(5.6% S)
Clean coal	80.3% with 9.3% "	(4.4% S)
Refuse	19.7% with 51.5% "	—

*Manufactured by the Roberts & Schaeffer Company, Chicago, Ill.
†Manufactured by the Stephens-Adamson Company, Aurora Ill.

MINING SOCIETY OF NOVA SCOTIA
DONALD J. MACNEIL, *President* S. C. MIFFLEN, *Secretary*

Coal-Fired Gas Turbines*

By T. E. WARREN, H. P. HUDSON, J. D. ROBERTSON, and T. R. SKERRY†

(*Annual Meeting, Mining Society of Nova Soctia, Sydney, N.S., July* 1952)

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INTRODUCTION

ALTHOUGH coal is generally the cheapest fuel on the basis of heating value it has been at a disadvantage in competition with other fuels because means have not been available for efficient conversion of its thermal energy to mechanical work. Taking the railway application as an example, the cost of Diesel oil is about three times that of its thermal equivalent in coal, but the overall efficiency of a Diesel locomotive is about five times that of a coal-fired steam locomotive. Accordingly, the Canadian railways are replacing all coal-fired steam locomotives, as they are retired from service, with Diesel units. This conversion in itself represents a major loss to the coal industry because the value of railway fuel is approximately 130 million dollars per year, and there are other applications in the medium power range where oil is displacing coal. There is, therefore, a strong incentive to attempt development of an efficient coal-burning engine.

Coal has never been a promising fuel for reciprocating internal combustion engines because of the time required for its combustion in pulverized form. This is not less than one-tenth of a second so that the speed of such an engine would have to be very slow. There are also serious difficulties of a mechanical nature arising from the necessity for intermittent combustion and the use of close-fitting pistons. Coal can, of course, be used indirectly by prior gasification, but the low calorific value per unit of volume of producer gas, the necessity for removing dust and tar from it, and the large size of the gas producer, have proved to be formidable difficulties. Recently, therefore, attention has turned to the gas turbine as a possible means of using coal for fuel in an engine of relatively high efficiency. In it, combustion is continuous, combustion time is not a critical factor, and compression and expansion are accomplished by rotating components, eliminating problems of lubrication and cylinder wear. It is apparent, therefore, that the gas turbine has none of the characteristics which precluded the use of coal as fuel for the reciprocating internal combustion engine. It should not be inferred, however, that the coal-fired gas turbine is already established, for, as will be shown, it presents numerous problems and not all of them have been satisfactorily solved.

The concept of the gas turbine is not new and attempts at practical development have been made over a period of about ninety years. It was not until 1939, however, that technical success was achieved. In that year the Brown Boveri Company completed installation of a 4,000-kw. oil-burning power generator at Neuchatel in Switzerland. In 1941 the same Company put a locomotive unit into operation on the Swiss Federal Railways. The war delayed progress of industrial applications but stimulated research on jet engines which led to improvements in turbo-compressors and in high-temperature alloys for turbine construction. These improvements were incorporated in the industrial units that have been built, at an accelerating rate, since 1946. By 1950, thirty-two different units had been built or were planned in the United Kingdom, there were twenty-three in Switzerland, twenty-two in the United States, fifteen in France, and one each in Denmark, Holland, Sweden, Japan and Canada.

Most of these were designed to burn clean distillate fuel oils but an increasing effort is now being applied to investigation of methods for utilizing residual oils and coal. This is because it is difficult for the gas turbine to achieve an efficiency as high as that of its principal competitor, the Diesel engine, and, if it must use the same type of fuel, it is at a disadvantage in regard to operating cost. It is, therefore, vital to the extension of the scope of gas turbines to find means for utilizing cheap fuels.

The cheap fuels to be considered — coal and petroleum residuums — are both handicapped by the presence of inorganic matter. This appears in the combustion products as ash which can damage the turbine or lower its performance by erosion, deposition, or chemical corrosion. Coal has the higher content of inorganic matter and is therefore expected to present the greater difficulties in regard to erosion and deposition. On the other hand, the ash from asphaltic oils has a high content of vanadium oxide which is known to have a severe corrosive effect on many alloys. This condition restricts the selection of materials for construction of turbines and heat exchangers and may also limit the turbine inlet temperature, which is the principal factor determining thermal efficiency. It may, therefore, prove to be no more difficult to use coal as a gas turbine fuel than to use residual oil. If this is found to be true, coal and bunker oil will compete on the basis of cost per unit of heating value and in most instances coal will have the overall advantage.

The four recognized ways of using coal as a gas-turbine fuel can be classified as follows:

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†Division of Fuels, Mines Branch, Department of Mines and Technical Surveys.

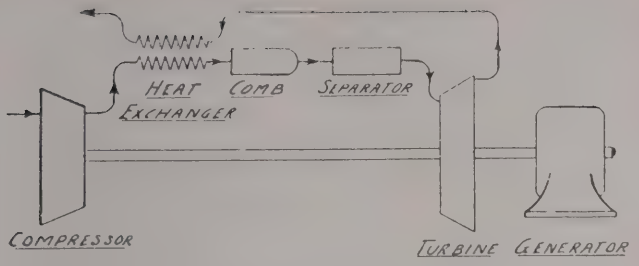


Figure 1.—Open cycle gas turbine with heat exchanger and ash separator.

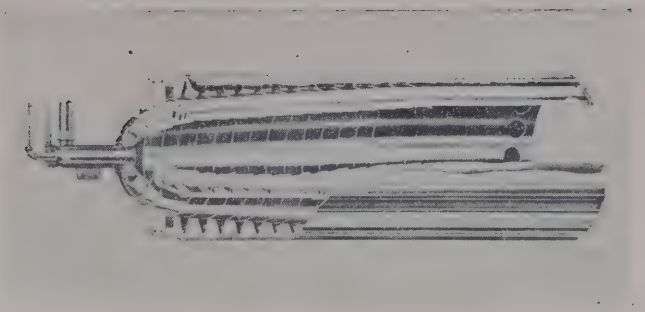


Figure 3.—Locomotive Development Committee combustor.

1.—Internal combustion:
- (*a*) Direct firing
- (*b*) Gasification of coal with direct firing of producer gas

2.—External combustion:
- (*a*) Closed cycle
- (*b*) Exhaust-heated cycle

Internal combustion by gasification of coal and direct firing of producer gas is the subject of an investigation sponsored by the Ministry of Fuel and Power of the United Kingdom. A 2,000-kw. unit is under construction and will possibly be in operation in 1952 (1). This cycle has not been selected for experimental study in Canada because it does not appear to be easily adaptable for the locomotive application. The closed-cycle turbine is also under investigation by the Ministry of Fuel and Power in collaboration with John Brown and Company, Ltd. (1). The necessity for a large quantity of cooling water, however, precludes its use in mobile equipment, so that it is of less interest in Canada than the two remaining cycles which are considered adaptable to railway locomotives. The Fuels Division of the Mines Branch has participated in projects for development of the direct-fired cycle and the exhaust-heated cycle. The experience so gained is the basis for the following discussion of the difficulties inherent to internal and external combustion cycles and the means that have been employed to overcome them.

(1) For references, see end of paper.

Internal Combustion

A diagram of the open cycle, equipped in this case with a heat exchanger and an ash separator, is shown in Figure 1. In this cycle, the fuel is burned in the stream of compressed gas between the compressor and turbine. The major difficulties in coal-firing such a cycle are immediately apparent. The first of these is that coal must be charged to the combustor against a pressure of about 60 pounds per square inch. Secondly, it must be burned, with high efficiency, under unusual conditions. Finally, the solid products of combustion must be removed before they encounter the turbine blading. The open cycle is the basis of the project of the Locomotive Development Committee of Bituminous Coal Research, Inc., and the methods they have employed to overcome these difficulties will be outlined.

The Committee has adhered to its original plan to fuel the locomotive with run-of-mine coal. Equipment for crushing, drying, and pulverizing is therefore to be carried on the locomotive. Some novelty of design has been necessary in order to meet space limitations and to control feed rates between components, but in general this equipment has followed established practice. At the point where the pulverized coal is introduced into the high-pressure system, however, a major problem is encountered.

After considering and experimenting with various apparatus, the Committee has developed the metering coal pump shown in Figure 2 (7). This is essentially a toothed rotor resembling a gear which rotates in a closely fitting casing. As each pocket passes the top position, pulverized coal from the storage tank falls into it and is transferred to the bottom position, where it is picked up by a stream of high-pressure air. The speed of rotation can be varied in order to control the fuel rate and so the coal pump acts in effect as the throttle for the turbine. The problem of sealing the rotor is, of course, formidable but has been solved by fitting the rotor

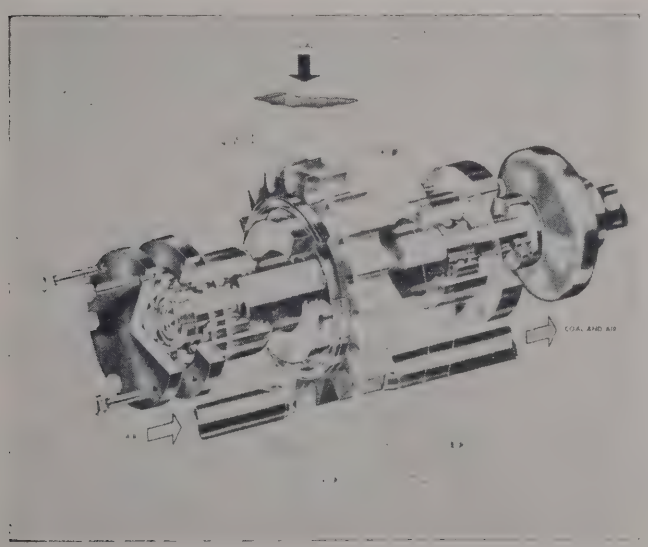
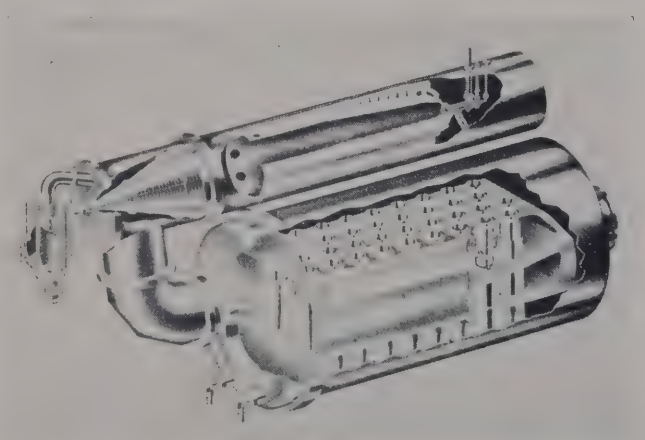


Figure 2.—Coal pump.



Figur. 5.—Locomotive Development Committee dust separating equipment.

Figure 4.—Locomotive Development Committee combustor installation.

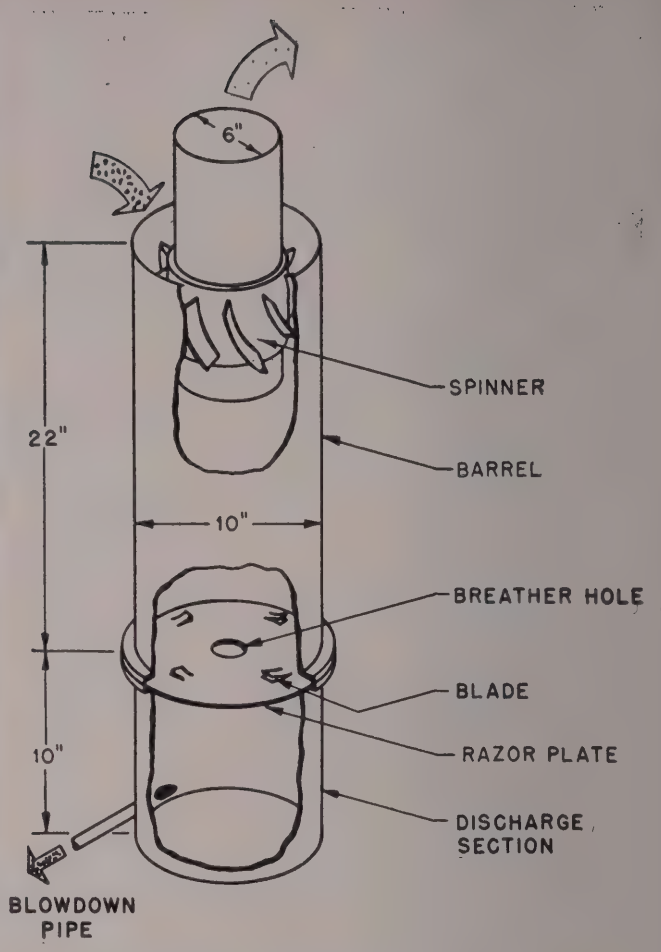


Figure 6.—Dunlab dust separator.

and casing to clearances of 3 to 5 thousandths of an inch and by employing a specially designed rubber sealing ring to prevent leakage in the radial direction. Erosion of the rotor has also presented some difficulty, but it is believed that this can be overcome by hard-surfacing. The unbalanced force on the rotor is offset by rigid construction of the shaft, precision ball-bearings, and great care in assembly. Pumps of this type have been used for 1,000 hours on the pilot scale and have operated more than 178 hours on the 4,000 h.p. full-scale unit. It has been concluded that the coal pump is a feasible means of introducing pulverized coal into a compressed-air line but that the design should be simplified so as to make it easier to build.

The next major problem to be considered is that of combustion. The function of the combustor is to burn a sufficient quantity of pulverized coal in a stream of compressed air to raise its temperature from about 660°F., at which it leaves the recuperator, to 1,300°F. for release through the turbine. The quantity of air to be heated is about six times the requirement for efficient combustion, so that most of the air must by-pass the flame zone and be mixed with the combustion products beyond it. Combustion must be nearly complete, not only in the interest of efficiency but also in order to prevent burning in the dust-collecting equipment. Space limitations on a locomotive make it desirable to have a heat release in the order of one million B.t.u. per cubic foot per hour. The Locomotive Development Committee decided to use an all-metal combustor because of its resistance to mechanical and thermal shock. The metal temperature, however, is limited to about 1,500°F. and, in order to prevent overheating and ash deposition, solid particles must not be allowed to impinge on the walls of the combustor.

The combustor that has been developed to meet these requirements is illustrated in Figure 3 (7). In it, coal and an equal weight of carrying air are admitted through a burner at the inlet end of the flame tube. Enough secondary air for complete combustion passes through and around the burner. The remaining air goes through the annular space between the flame tube and the outer shell and part of it passes through slots formed by the overlapping rings of the flame tube to produce a film of relatively cool air along the inside of the tube. The final addition of air is at the end of the flame tube for the purpose of quenching the products of combustion to a uniform temperature of 1,300°F. This type of combustor has performed well in the pilot plant for more than 1,000 hours and twin combustors of the same general design have been used for more than 178 hours of testing at full scale.

The greater part of the work of the Locomotive Development Committee was conducted on Ohio No. 8 seam coal, which is classified as high-volatile *A* bituminous. It was considered that a systematic study should be made of the effect of variation in coal properties on combustion performance. This was undertaken for the Committee by the Fuels Division of the Mines Branch (5). Performance and efficiency tests were run on 23 coals of various groups within the bituminous class of the A.S.T.M. classification by rank. The tests were made with a combustor of the type and size illustrated in Figures 3 and 4. They differed from actual operation with the gas turbine in that they were made at atmospheric pressure. Basic research made for the Locomotive Development Committee by Battelle Memorial Institute had shown that the rate of combustion of pulverized coal did not

vary significantly with change in pressure and it was considered, therefore, that the comparative tests at atmospheric pressure would be applicable in the range of pressures up to 60 pounds per square inch.

The tests showed a major variation in combustion efficiency with change in rank. Observed efficiencies varied from 44 to 90 per cent and in general were in inverse order of rank. The results are summarized in Tables I and II.

Somewhat higher efficiencies have been attained by the Locomotive Development Committee when operating at full cycle pressure with a different burner. It is possible, therefore, that the results observed by the Fuels Division do not apply generally to this type of combustion but are peculiar to the conditions under which they were obtained.

It is after the coal has been pumped into the high-pressure system, burned, and the products of combustion quenched to 1,300°F. that the most critical difficulty is encountered. Large ash particles erode the turbine blading and must be removed before they reach the turbine. The problem is therefore to devise separating equipment that will remove substantially all particles of diameter greater than 20 microns (0.00079 inch) and a major proportion of those greater than 10 microns. The separator must be small enough to install on a locomotive and must operate with a small pressure differential. Provision must be made to remove very large pieces of slag, scale, or tramp metal which tend to bounce or sail through equipment that removes much smaller particles.

When the Locomotive Development Committee undertook the project there was no dust separator that could meet these conditions and a new design had to be developed. The equipment that was finally employed on the full-scale unit is illustrated in Figure 5 (7). This unit has twin combustors, only one of which is shown in the illustration. Referring to Fig. 5, the combustion products are first led through a conical louvre separator in order to remove large pieces of slag or other objects that are carried in the stream of gas. The gas then goes through 26 specially designed 'Dunlab' separators arranged in parallel, to remove the smaller particles.

The Dunlab separator is shown in Figure 6. The gas to be cleaned enters the annular space at the top and is given high rotational velocity by the spinner vanes. Dust is thrown against the walls of the barrel and passes downward through narrow apertures under a row of thin 'razor blades' into the discharge section of the tube. From the bottom of the discharge section it is removed in a small stream of air through the blow-down pipe. Cleaned gas passes up the centre of the barrel to the outlet tube at the top.

The design of the Dunlab separator was probably the most important achievement of the whole project, as is illustrated in Figures 7 and 8. Figure 7 shows the degree of erosion of the stator blades of the turbine after 250 hours of operation with a previous design of separator tube which allowed the passage of ash comprising 41 per cent of particles larger than 10 microns and 6.6 per cent larger than 20 microns. Figure 8 shows blading in the same position in the turbine after

TABLE I.— AVERAGE EFFICIENCIES OF COALS TESTED

COAL NO.	COAL RANK	AVERAGE EFFICIENCY (per cent)
1	High-volatile bituminous C	89.9
2	High-volatile bituminous A	88.8
3	High-volatile bituminous B	86.7
4	High-volatile bituminous C	85.6
5	High-volatile bituminous A	78.6
6	High-volatile bituminous A	77.7
7	High-volatile bituminous B	77.1
8	High-volatile bituminous A	76.8
9	High-volatile bituminous A	76.7
10	High-volatile bituminous C	74.6
11	High-volatile bituminous B	73.1
12	High-volatile bituminous A	72.7
13	Medium-volatile bituminous	66.0
14	Medium-volatile bituminous	65.6
15	Medium-volatile bituminous	58.5
16	Medium-volatile bituminous	58.3
17	Medium-volatile bituminous	57.9
18	Medium-volatile bituminous	57.7
19	Medium-volatile bituminous	54.3
20	Medium-volatile bituminous	48.1
21	Medium-volatile bituminous	46.0
22	Low-volatile bituminous	45.9
23	Medium-volatile bituminous	43.6

TABLE II.— AVERAGE EFFICIENCY FOR ALL COALS OF GIVEN RANK

RANK	AVERAGE EFFICIENCY FOR ALL COALS OF GIVEN RANK (per cent)
High-volatile bituminous C	83.3
High-volatile bituminous B	79.0
High-volatile bituminous A	78.5
Medium-volatile bituminous	55.6
Low-volatile bituminous	45.9

Figure 7.—Eroded stator blades.

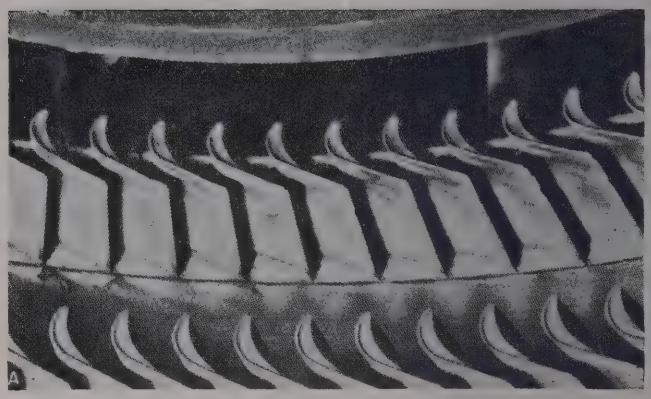


Figure 8.—Stator blades used with Dunlab separator.

operation for the same length of time with the Dunlab separator, which allowed passage of ash with only 16 per cent larger than 10 microns and 2 per cent larger than 20 microns.

It is apparent from the foregoing discussion that the three major difficulties inherent in the open cycle can be overcome. The Locomotive Development Committee has succeeded in operating the first coal-fired gas turbine for periods totalling 1,000 hours (7). They have also constructed a 4,000 h.p. unit within the space limitations of locomotive design and have operated it for 178 hours. This unit is currently undergoing a 750-hour test (8, 9). Although this is a highly creditable achievement and it is probable that the technique can be improved, the exhaust-heated cycle also merits full experimental investigation.

External Combustion

The exhaust-heated cycle has been described in a paper published by Professor D. L. Mordell of McGill University in 1949 (2). In this cycle, shown in Figure 9, air is compressed and passes through a heat exchanger which brings it to the required turbine inlet temperature. After the air has passed through the turbine, fuel is introduced and the temperature raised for passage through the low-pressure side of the heat exchanger. This cycle has the advantages that only clean air is sent through the compressor and turbine, and combustion may be conducted at low pressure. It is not necessary, therefore, to pump coal into the high-pressure system or to take any precaution to protect the turbine blading from ash. On the other hand, the thermal efficiency is reduced, the heat exchanger presents unusual problems, and the combustor, if fired with pulverized coal, would have to be very large.

The thermal efficiency of the simplest form of exhaust-heated cycle is lower than that of the internal combustion cycle because all of the heat liberated in the combustor cannot be transferred by the heat exchanger to the compressed air. In order to maintain the required rate of heat transfer there must be an adequate temperature difference between the hot and cold streams in counter-current flow in the heat exchanger. The heat capacities of the two streams are approximately the same, so that the hot gas must be discarded at a temperature much higher than that of the compressed-air stream entering the exchanger.

The loss of efficiency can be almost entirely corrected by an improvement that has been devised since publication of the first paper on the exhaust-heated cycle (3). The improved cycle is shown diagrammatically in Figure 10. It will be seen that the turbine exhaust is divided into two parts. One of these passes through the combustor and is raised in temperature by combustion of fuel. It then passes through a heat exchanger where its temperature is reduced to that of the initial turbine exhaust. The two parts of the turbine exhaust are then united and conducted through a second heat exchanger. The advantage of this arrangement is that the heat of combustion is donated to a relatively small quantity of air, bringing it to a high temperature. As a consequence, it is possible to transfer all of the heat of combustion to the compressed air. The net thermal effect is therefore the same as if the fuel had been burned directly in the compressed-air stream, and the thermal efficiency of the improved exhaust-heated cycle is equal to that of the internal combustion cycle if the pressure drop through the hotter heat exchanger is neglected.

The improved cycle, however, presents a new problem. The gas entering the hot end of the hot heat exchanger is at a temperature of nearly 2,000°F. The heat exchanger must, therefore, be constructed in part of heat-resistant alloys and even with them some novelty of design is required to keep the metal temperatures as low as possible. The design that is to be used on the prototype engine now under con-

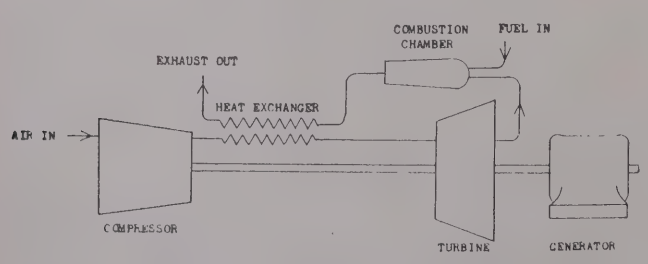


Figure 9.—Exhaust heated cycle.

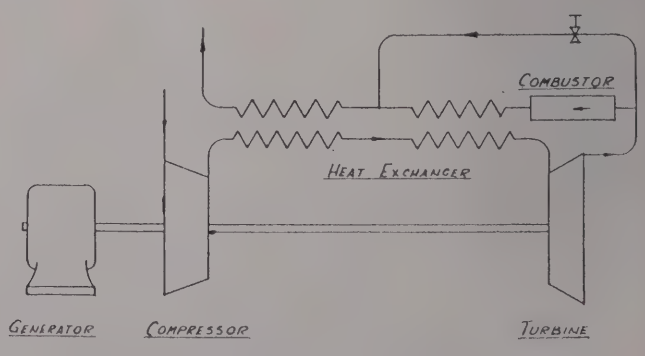


Figure 10.—Improved exhaust heated cycle.

struction uses counter-flow of air and combustion gas at the cooler end of the exchanger and parallel flow at the hotter end, which results in lower maximum metal temperatures. Special design features are also necessary to prevent overheating of the tube sheet. These have not yet been tested in practice and will be described in a later publication when test results are available.

In addition to the design problems introduced by high temperature, consideration must be given to the possibility of fouling of the heat exchangers with ash. Part of the ash in the fuel must pass through the exchangers. For this reason, the combustion gases are conducted inside the tubes, and the clean air outside them, so that the surfaces exposed can be made easily accessible for cleaning.

The effect of tube fouling on the rate of heat transfer is an important factor in design of the heat exchanger but is not calculable. An experimental study of tube fouling was therefore made in the laboratories of the Division of Fuels at Ottawa. Conditions in the hot exchanger of the prototype engine were simulated in an exchanger with four one-inch tubes ten feet long. Tests were made over 35-hour periods with five different coals. The ash remaining in the tubes was found to be non-adherent and could be removed by light brushing. In no case was its amount greater than 2 per cent of the ash in the coal charged. Throughout the 35-hour tests the overall heat transfer coefficient from combustion gas to air was also measured. The results of a test of 92 hours' duration are shown in Figure 11. It will be observed that, over the period of the test, the heat transfer coefficient did not decrease. These tests give some reassurance that ash fouling of the heat exchanger tubes will not be a critical factor in performance of the exhaust-heated turbine, but final conclusions can be drawn only after prolonged operation of the prototype engine that is now under construction.

The function of the combustor in the improved exhaust-heated cycle is to raise the temperature of part of the air exhausted from the turbine from about 900°F. to about 1,900°F. The quantity of air passing through the combustor and hot heat exchanger is about one-half of the total cycle air, and both the absolute quantity and the proportion of the total may be varied in order to control the power output of the turbine. The air traversing the combustor circuit is about three

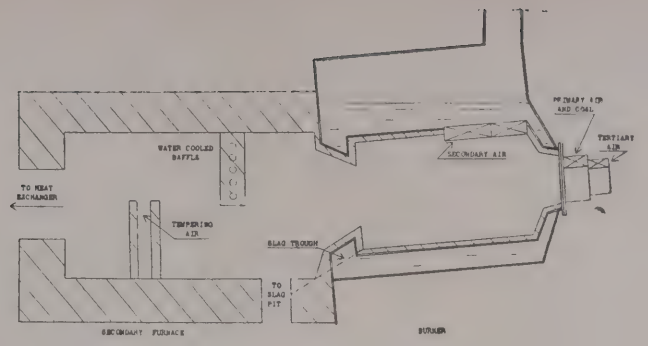


Figure 12.—Diagram of cyclone combustor as adapted for prototype coal fired gas turbine.

times the stoichiometric requirement for combustion and, in order to avoid inefficient operation with 200 per cent excess air, about half of it must bypass the combustor and be added between it and the hot heat exchanger. The pressure in the combustor is only about 1.5 pounds per square inch above atmospheric pressure.

In addition to these conditions imposed by the nature of the cycle, other features of the combustor are limited by the locomotive application and by the necessity to prevent fouling of the heat exchanger tubes. A 4,000 h.p. locomotive operating at full power will require a heat release of about 50 million B.t.u. per hour. A high rate of heat release, in the order of one million B.t.u. per cubic foot per hour, is therefore required so as to keep the combustor dimensions within permissible limits. Such a rate is not attainable with pulverized coal firing at atmospheric pressure. Firing of lump coal on grates would require special materials or technique because of the high preheat temperature of the primary air. It is also desirable to prevent as much as possible of the ash in the coal from passing on to the heat exchanger in the form of fly ash.

The basic principle of the cyclone burner developed by Babcock and Wilcox appears to fulfill the combustor requirements to a higher degree than any other (6). This type of burner is cylindrical in shape and is operated at such a temperature that a film of molten ash covers its interior surface. Crushed coal, carried in a stream of primary air, is introduced tangentially at high velocity and is thrown into the molten ash film by centrifugal force. Secondary air is also introduced tangentially and flows past the embedded coal particles at high velocity, so that very rapid combustion takes place. The furnace is inclined downward toward the exit end at an angle of about 10 degrees to facilitate flow of ash to a collecting sump.

Furnaces based on this principle have operated successfully in stationary power plant applications. They are especially adaptable to coals having a high content of low melting point ash. About 80 per

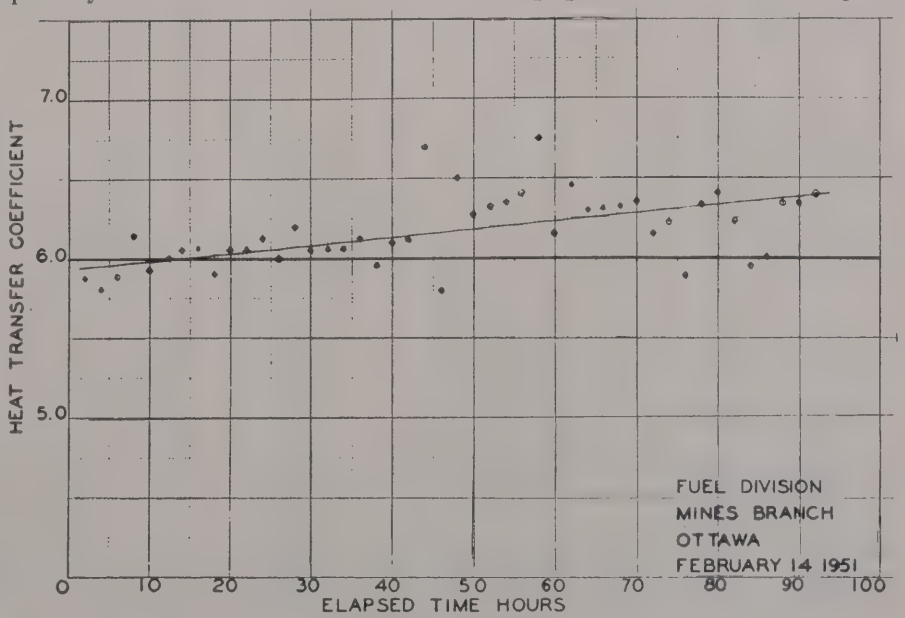


Figure 11.—Heat transfer coefficient in experimental heat exchanger under simulated conditions of operation.

cent of the ash is recovered as slag. The coal feed is crushed to pass a No. 4 U.S.S. sieve which has an opening 0.187-inch-square and requires no pulverizing. Heat releases of 4 to 5 hundred thousand B.t.u. per hour were obtained in tests on the large stationary units. Combustion efficiencies were high; the losses as unburned carbon were 0.01 to 0.13 per cent and there was no measurable loss as carbon monoxide. This type of combustor has been demonstrated, therefore, to have many of the characteristics required for use with an exhaust-heated cycle locomotive.

Means for adapting the cyclone principle for gas-turbine use are being investigated by the Fuels Division of the Mines Branch. Several small experimental combustors have been built and tested. These have demonstrated that heat releases of the order of one million B.t.u. per cubic foot per hour are attainable with combustors up to two feet in diameter. The combustor developed for the prototype engine is shown diagrammatically in Figure 12. The combustion chamber is a water-cooled steel cylinder lined with rammed plastic refractory which is held in place and cooled by steel studs welded to the interior surface of the cylinder. Following the combustor is a secondary furnace containing the slag pit, baffles, and port for admission of tempering air.

One difficulty that has been encountered in operation of this combustor is entrainment of slag in the air stream passing through the secondary furnace. The effect of this entrainment is shown in Figure 13 where it will be noted that an accumulation of slag has formed over the tempering air inlet. In order to correct this entrainment, the secondary furnace has been enlarged so as to reduce the gas velocity, and a trough has been run through the combustor lip to the slag pit. Tests with the modified combustor are now in progress. Photographs of the cyclone combustor installation are shown in Figures 14 and 15.

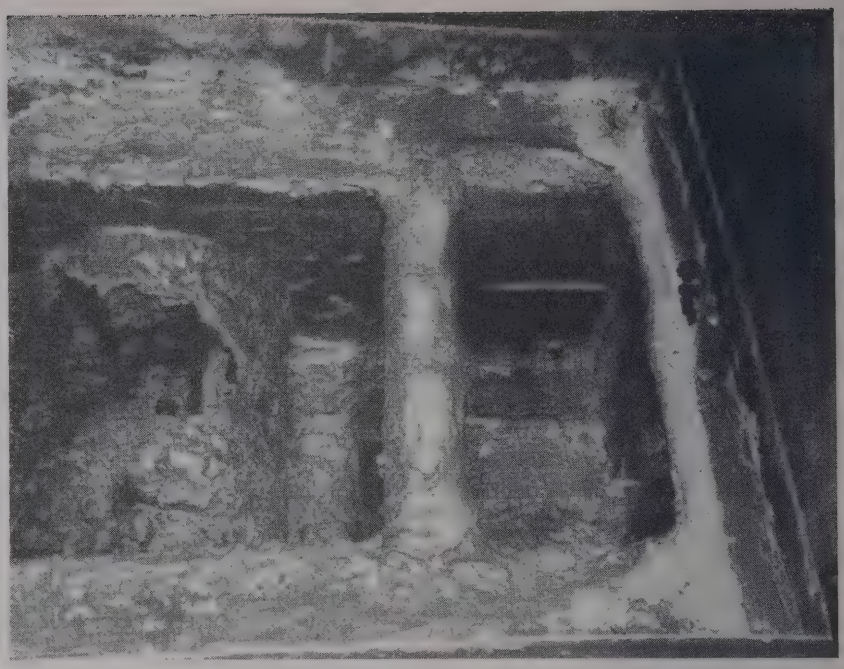


Figure 13.—Slag formation in secondary furnace of cyclone combustor.

Development of the exhaust-heated cycle has not reached the same stage as that of the open cycle of the Locomotive Development Committee. The prototype engine that has been referred to is being built in the Gas Dynamics Laboratory of McGill University as a joint project of the University and the Mines Branch (4). Construction is well advanced but no definite time schedule for operation can be given. Final assessment of the new design features of the heat exchanger and combustor can be made only after test results with the prototype engine are available. The exhaust-heated cycle is also to be tested in the United Kingdom, where the Ministry of Fuel and Power has commissioned manufacture of an 1,800 h.p. locomotive by C. A. Parsons and Company (1).

Conclusions

From the foregoing outline it is apparent that the coal-fired gas turbine is at present only in the development stage. There is no sound basis for predicting the time when it will be in extensive commercial use. It is not certain that it will ever reach that status. Some reassurance can be drawn, however, from the fact that no technically insurmountable difficulty has yet been recognized. Moreover, the interests of coal producers, railways, and turbine manufacturers, as well as general considerations of economy and security,

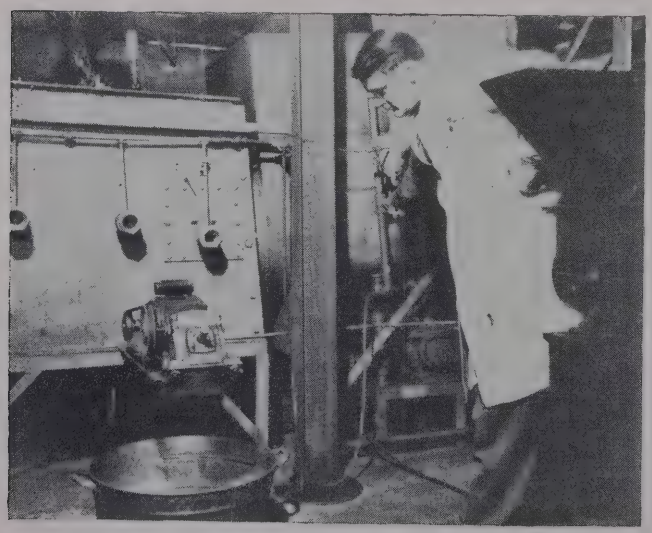


Figure 14.—Cyclone combustor.

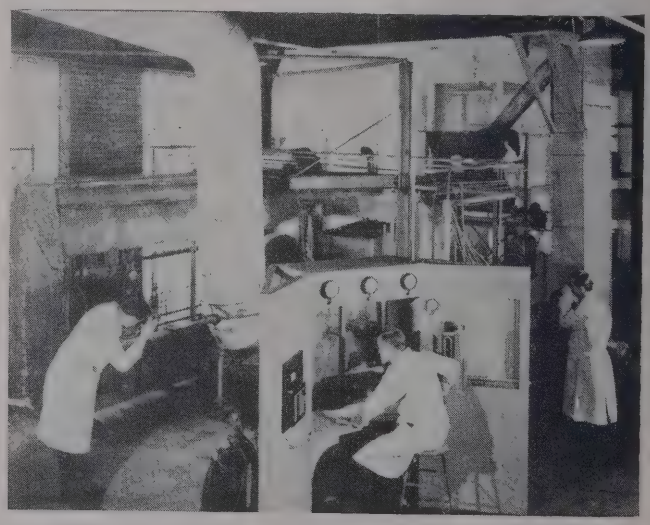


Figure 15.—Cyclone combustor installation.

combine to create a strong incentive to pursue the development. It is probable, therefore, that development will continue until a satisfactory coal-fired gas turbine is achieved or is known, for some definite reason, to be unattainable.

Acknowledgments

The writers are indebted to Mr. John I. Yellott for permission to use material and illustrations from reports of the Locomotive Development Committee of Bituminous Coal Research, Inc., and to Professor D. L. Mordell for advice and criticism regarding the present paper.

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MINING SOCIETY OF NOVA SCOTIA
DONALD J. MACNEIL, *President* S. C. MIFFLEN, *Secretary*

Recovery of Barytes at Walton, N. S.

By G. G. CAMPBELL[*]

(*Annual Meeting, Mining Society of Nova Scotia, Sydney, N.S., July,* 1952)
(*Transactions, Volume LV,* 1952, pp. 316-319)

HISTORICAL

BARYTES was mined in Nova Scotia as early as 1874 from a deposit at Five Islands, on the north shore of Minas Basin. The first record of the presence of the mineral at the Walton site appears in a report by Selwyn in 1893, who states: "Barytes in considerable quantity occurs in veins in limestones at Walton and Pembroke". The exact site of the present workings was marked on a map published by Fletcher in 1905.

The economic possibilities of this deposit, however, were not recognized until 1940, at which time we were at war with Germany, who up to that time had supplied 50 per cent of the world's requirements.

At that time the Nova Scotia Department of Mines, recognizing the need for certain strategic minerals, induced Canadian Industrial Minerals, Limited, to send prospectors into the area to look for manganese. No manganese deposits were found, but a deposit of barytes was directed to their attention, and exploration and development work on the occurrence was proceeded with.

INTRODUCTION

The south shore of Minas Basin between the Avon river at Windsor and the Stewiacke river at Maitland has long been known as a potential source of mineral wealth. It had attracted the attention of mineralogists and in the past had produced considerable tonnages of manganese ore and gypsum. Depletion of ore in surface and near surface deposits resulted in a dwindling production, until today only deposits of gypsum are worked.

[*]Manager, Canadian Industrial Minerals, Limited.

The region is heavily drift covered, making prospecting exceedingly difficult and costly, and limiting the operations of the ordinary prospector. Outcrops of bed-rock are few and scattered, being confined almost entirely to the sea shore and the floors of the small, sharp valleyed, streams. The general rock structure is indicated by the areal geology, but detailed structures which can be expected to control ore depositions are hidden by the drift and require subsurface evidence to delineate.

GEOLOGY

The regional geology of the area studied by Logan, Selwyn, and Fletcher was first compiled into a map by Fletcher in 1905. The whole region is heavily drift covered, recent drill records showing thicknesses up to 90 feet of glacial gravels and tills. Bed-rock consists of Carboniferous limestone, sandstone, and shales, for the most part stained with iron and thus showing arid conditions of deposition. Gypsum beds are not infrequent, and several saline springs are scattered through the area. Triassic sandstones and conglomerates, with remnants of amygdaloidal flows and cut by basic dykes, are exposed close to the shore of Minas Basin. The geological time-table is shown in Table I.

The Windsor-Horton contact appears to be the most important or significant geological feature of the area. As a consequence of the general arid condition at the time of deposition there was a widespread distribution of iron throughout the sediments. At the same time, earth movements brought about rapid changes, both in the extent of the sea and in the character of the sedimentation in the sea. Horton sandstones are replaced by shales and shaly limestones in rapid succession, both vertically and laterally. At the beginning of Windsor time, the seas cleared somewhat, but embayments were cut off by land changes and evaporation from these bays resulted in precipitation of calcium carbonate and calcium sulphate from the saturated sea waters.

The general structure of the region is a broad syncline, striking northeast and pitching to the east. The axis of the syncline is well south of the area under consideration, so that, in the vicinity of the barytes deposit, the bed-rock trends

TABLE I.—TABLE OF FORMATIONS

ERA	PERIOD	FORMATION	LITHOLOGY
CENOZOIC	Recent Pleistocene	River and lake deposits Ground moraines	Silts and sands Gravels and boulder clays
		Uncontormity	
MESOZOIC	Triassic	Intrusives and extrusives Sandstones and conglomerates	Lava flows, basalt and diabase dykes, etc.
		Unconformity	
PALAEOZOIC	Carboniferous	Windsor Horton	Sandstone, shale, limestone, gypsum Shale, sandstone, conglomerate

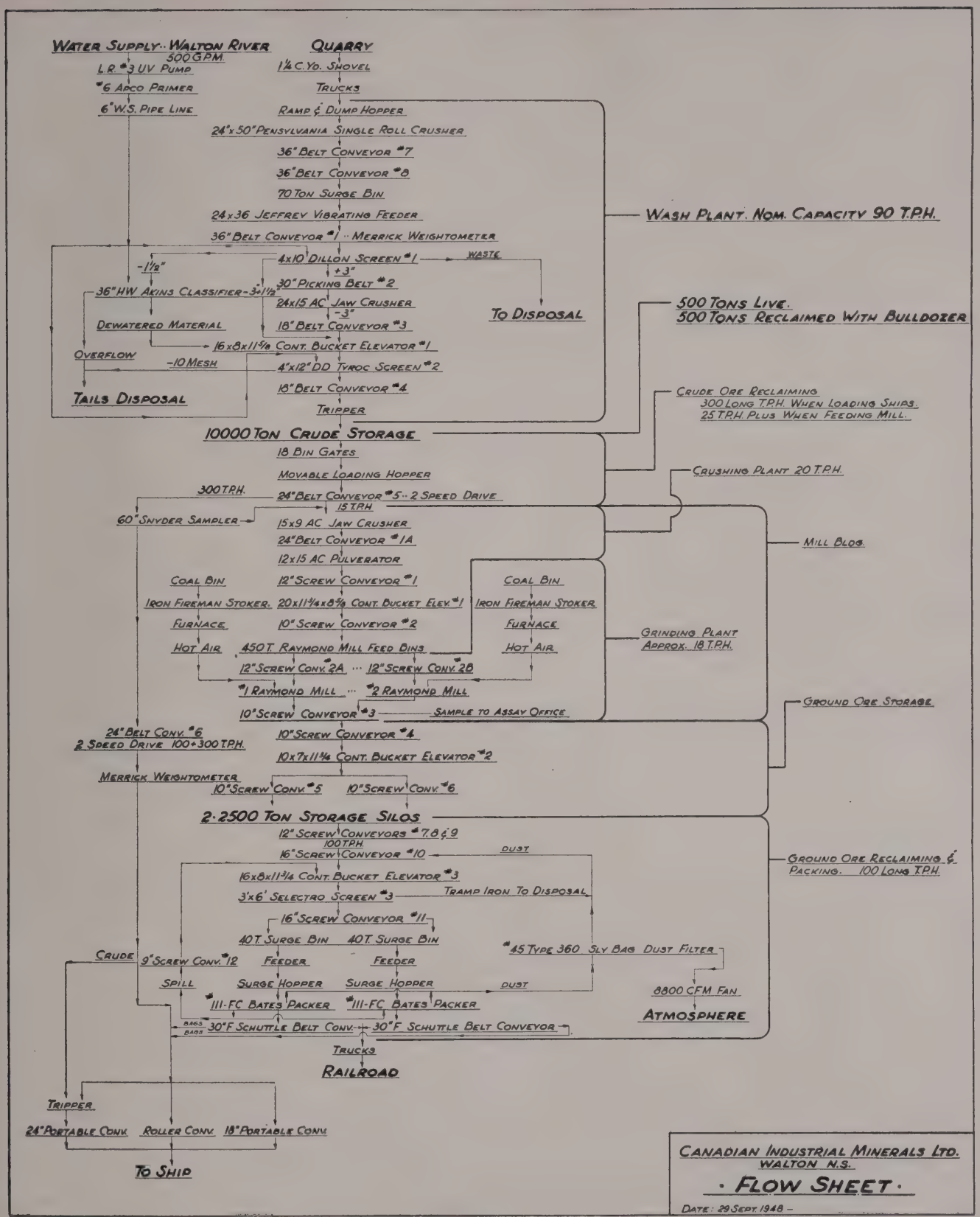

Figure 1.

northeasterly, with dip to the south.

Local changes in dip are common and there are undoubtedly many minor structures superimposed on the major structure, but these have not as yet been worked out in detail

The primary structures are presumably due to broad earth movements that occurred at the close of the Permo-Carboniferous period. The secondary structures may be contemporaneous, or, and more probably are, the result of movements in Triassic time, associated with lava flows and basic intrusions that make up North mountain of the Annapolis valley and the similar rocks exposed along the north shore of Minas Basin. The mineral deposits, both manganese ore and

barytes, appear to be replacements of certain beds or portions of beds by percolating solutions, but whether they were of supergene or hypogene origin remains to be proved. One is inclined to think that the manganese and barium had their source in the Triassic lavas and basic intrusions, although they may have been transported to their present positions by surface waters.

Market Specifications

Market specifications for barytes depend on the purpose for which it is to be used.

(1) *Chemical Grade.* — This grade, produced from the higher grade ores, is used as a principal ingredient in the manufacture of lithopone, a synthetic pigment for paints, fillers for linoleums, etc. The specifications for this grade are as follows:

$BaSO_4$ content	95.00%	min.
Fe_2O_3 "	1.75	max.
SiO_2 "	2.00	"
Al_2O_3 "	0.50	"
CaO "	0.75	"
$SrSO_4$ "	2.25	"
Moisture	2.00	"

(2) *Oil-Well Grade.*—This material is used as a component in circulating drilling fluids. The finely ground barytes (minus 325 mesh) is used with bentonite and a dispersing medium in water suspensions in drill holes. The heavy fluid so formed, weighing about 120 lb. to the cubic foot, acts both as a liquid seal against gas and oil pressures encountered when the oil sands are pierced by the drill, and as a means of removing rock cuttings from in front of the rotating bit of the drill. The specifications for barytes for this use are as follows:

Screen analysis	98.0% −325 mesh
$BaSO_4$ content	93.0% min.
Specific gravity	4.2% min.

Mining

The deposit outcropped originally on the crest of a low ridge and was exposed for a few square feet only. Trenching and pit digging outlined an area of 200 square feet over which the barytes was at a depth of 5 feet or less from the surface. Systematic diamond drilling on a grid pattern proved that the orebody was of considerable size, and indicated fairly definitely its trend and general outline. Various methods of mining the orebody were considered and opinion at first leaned toward some form of block caving. Subsequent diamond drilling and further development work disclosed that this method was ruled out for three principal reasons:

(1) The orebody did not have the height necessary for a block caving system.

(2) It had a very weak hanging-wall, consisting of non-competent shales and sandstones, above which was a very heavy overburden of sand and clay.

(3) Grade control in the mine would be impossible with a caving system.

The method finally decided upon was a combination of quarrying and glory-hole mining, following stripping of the earth and barren rock over the orebody. This method has the advantage of permitting 100 per cent recovery of the ore mined, it is extremely flexible, and it allows close inspection and selection of the ore in all phases of the operation. Recovery, dilution, and contamina-

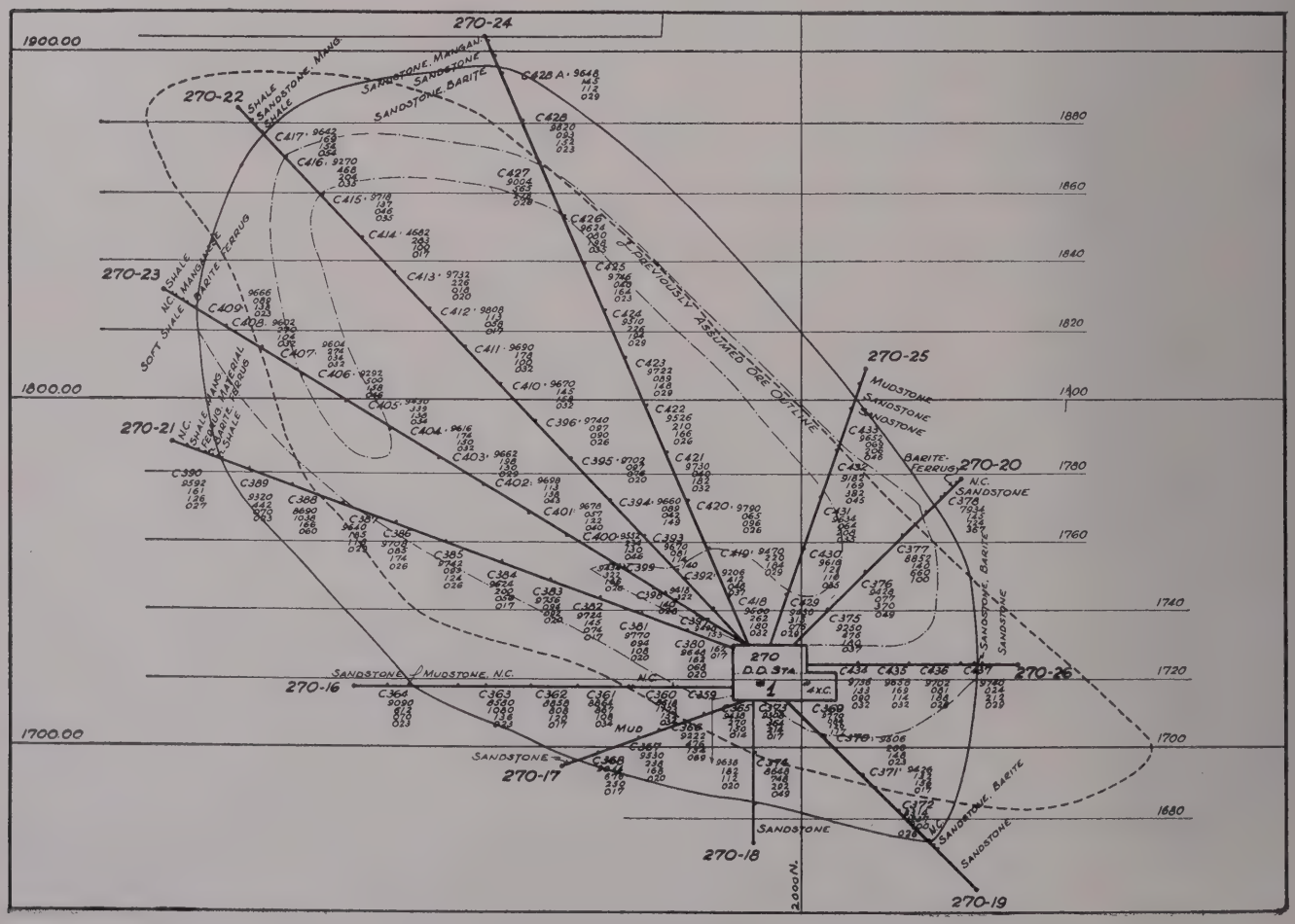

Figure 2.—Section through 270 No. 1 d.d. station.

Scale: 1 in. = 55 ft.

tion of the broken ore can be controlled to a much greater extent than with block caving. The system is working successfully and the hazards inherent in an untried mining method are not present.

In the quarry proper, it was necessary to remove 1,200.000 cubic yards of overburden and 800,000 cubic yards of rock in order to expose the ore.

For this work the Company purchased *Model C* LeTourneau roadsters, complete with *E-16* 'carry-all' scrapers with a capacity of 16 tons, or 13.5 cubic yards, and powered with Cummins *HRB-600* Diesel engines. The 1,200,000 cubic yards of clay and overburden was removed at a cost of 14 cents per cubic yard, including maintenance, but not depreciation and write-offs. The soft shales and sandstones were loaded by the carry-all scrapers with the use of a *K-30* LeTourneau rooter. The rooter was hauled over the shales and sandstones by a *D-8* caterpillar tractor, loosening the ground and rendering it suitable for loading with the scrapers. The rooting of this material added 4 cents per cubic yard to the cost but was found much cheaper than drilling and blasting. All of the carry-all scraper work was push-loaded by a *D-8* caterpillar tractor.

The harder rock making up the balance of the cap-rock covering the orebody is drilled and blasted, loaded into rock wagons and trucks, and hauled to the dumps. The drilling is done with Canadian Ingersoll-Rand *DB-30* rock drills mounted on wagon jacks. The holes are drilled to a depth of 20 feet. They have a 5-foot burden on the hole and are spaced at approximately 10-foot intervals. Flat holes are drilled in the bottom of the bench to give a better floor for the trucks and shovels. The cost of this rock work is currently running at approximately 50 cents per cubic yard.

A shaft was sunk to a depth of 350 feet and levels were established at 270 feet and 350 feet. From these levels raises were driven up through the orebody to the bottom of the quarry at strategic points to be used as mill holes. Chutes were installed in the raises and equipped with air-operated gates from which the ore can be drawn and hauled to the shaft and hoisted to surface.

In the quarry, the ore is mined by a bench system, the benches being carried at 20-foot intervals. The method of drilling is similar to that for the cap rock. The powder used is Dynamex, Stopelte, and occasionally 40 per cent Polar Forcite. Powder consumption is 1 pound of explosive for 4 tons of ore.

The broken ore is loaded by a North-West shovel into trucks of 3-ton capacity and hauled from the quarry to the wash-plant end of the milling plant, a distance of approximately three miles.

Milling and Shipping

The mill operates quite simply. Six electrically-operated processes do the job. These are washing, crushing, classifying, grinding, pulverizing, and bagging.

The ore is dumped from trucks into a chute and fed to a crusher. Here it is reduced to minus 5½-inch and is then taken by bucket elevator to a surge bin, from where it is fed onto a conveyor belt and carried to the wash plant. The ore is next carried by bucket elevator and belt conveyor to a 10,000-ton crude-ore storage shed.

Ore is taken from the crude storage shed through 18 draw points and carried by means of a truck loading hopper to a tunnel conveyor belt, which carries it either to a Raymond mill for fine grinding or direct to vessels loading crude ore.

When loading a vessel with crude ore a discharge trap door on the tunnel conveyor is closed and the ore, instead of going to the Raymond mill, is routed to a dock conveyor, passing, on its trip, over a weightometer which measures the load being put aboard a vessel. At the end of the dock conveyor, a hand-propelled track tripper transfers the ore to a portable conveyor which carries it across the dock and drops it directly into the holds of the vessels.

When feeding the ore to the Raymond mill for fine grinding it is carried from the tunnel conveyor to a crusher where it is reduced to minus 1-inch and then taken by a conveyor belt to the pulverator, in which it is further reduced to minus ¼-inch. This product is dumped into a bin from where it is automatically fed to the Raymond mill, which does the fine grinding. As fast as the ore is reduced to powder, the fine particles are drawn out by an air classification system. This system also throws back any oversize, allowing only the powder to be carried through. If the ore is wet, the air is heated before entering the classification system. By this means the moisture is evaporated and drawn off.

From the mill, the powder is carried by screw conveyors to either of two 2,500-ton cement storage silos.

When a vessel arrives at the dock for a cargo of ground ore, the powder is drawn from the storage silos by screw conveyors into bins feeding two bagging machines. Each bagging machine handles four bags and a skilled operator can fill as many as twenty bags per minute. By means of counter weights and adjustable stands, 50-, 100-, or 112-pound bags may be filled, as required. The filled bags from each bagging machine drop onto shuttle conveyors. These feed the bags direct to the dock conveyor from where they are transferred to the portable conveyor feeding one of the holds of the vessel. By manually transferring the bags to a second portable conveyor, two holds of the vessel can be loaded at the same time.

The portable conveyors discharge the bags to chutes leading to benches in the holds. The bags are picked from the bench by the stevedores and stowed in tiers.

A Reservoir Study of the Excelsior Pool

*By W. A. BAIN**

(*Annual Meeting, Petroleum and Natural Gas Division, Edmonton, Alta., May,* 1952)

(*Transactions, Volume LV,* 1952, pp. 320-325)

ABSTRACT

Production from the Excelsior Reservoir has amounted to approximately one million barrels. This production has caused a pressure drop of 10 p.s.i. from the original pressure of 1,245 p.s.i.g. Volumetric balance calculations indicate that the pool is being produced by a water drive whose strength is probably unsurpassed in Western Canada. It has also been shown by 'unsteady state' calculations that the reservoir will still be above the saturation pressure of 915 p.s.i.g. at the time of depletion. The average permeability of the producing formation is approximately 1,000 millidarcys and, consequently, well draw-downs are very low. The field is essentially a bottom-water drive and calculations have shown that gravity drainage will greatly assist the recovery of oil. Water flooding data on several cores have placed the flushing efficiency at approximately 50 per cent, which indicates a recoverable oil-in-place of 17.7 million stock tank barrels. Calculations and field data indicate that the MER for the early life of this field is probably as high as 8,000 to 10,000 stock tank barrels per day.

Introduction

THE EXCELSIOR NISKU oil reservoir is located about twenty miles northeast of Edmonton. The field was discovered in November, 1949, by Imperial Excelsior No. 1 well, in Lsd. 13, Sec. 3, Twp. 56, R. 22, W. 4, at a depth of 3,850 feet.

Since discovery, development has continued on a 40-acre spacing pattern. At the end of 1951, drilling over approximately 80 per cent of the estimated productive area of 1,680 acres had established 32 wells capable of production. The field has been fairly well delineated by dry holes on all but the eastern flank .

General completion practice in this field is to set casing through the producing formation and perforate the desired interval. An acid treatment of 1,000 to 2,000 gallons of 15 per cent HCl is used to clean the perforations and to improve the permeability of the adjacent formation.

Total oil production to December 31st, 1951, was approximately one million barrels. Water production has amounted to less than one per cent of the total fluid withdrawals. Six edge-wells in the field have been equipped with pumps, and none of these is making excessive amounts of water at this time.

Full-diameter diamond cores have been obtained from ten wells. These cores have been analyzed for porosity and permeability values and some representative plugs have also been subjected to water flooding.

The estimated 17.7 million barrels of recoverable oil-in-place do not classify Excelsior as a major oil reservoir, but, none the less, it is very important from a reservoir engineering standpoint because of its exceedingly strong water drive. This paper outlines the results of a study of the probable effects of this water drive on the ultimate oil recovery.

Geology

Production in the Excelsior reservoir is from the Nisku (*D*-2) member of the Winterburn formation, of Upper Devonian age. The reservoir rock is a gently dipping coralline biostrome which extends for many miles in all directions. The reservoir trap is structural and was caused by differential compaction of the green shales over the underlying Leduc (*D*-3) bioherm.

The Nisku zone is approximately 160 feet thick in the Excelsior area. The reservoir rock is a buff, crystalline, dolomitized limestone with a certain amount of anhydrite, silt, and argillaceous in-filling. The silty material is found near the crest of the structure while the dolomite grades into shale near the base. Good porosity is generally encountered throughout the entire section, with best development being found in wells near the crest of the structure.

Reservoir Characteristics

A summary of basic data for the reservoir is given in Table I.

Average depth of production is approximately 3,850 feet, or 1,550 feet subsea. A contour map of the top of porosity (Figure 1) clearly shows that the reservoir trap is structural and that none of the field is delineated by porosity pinch-outs. Figure 2 is a contour map of the water-oil contact. All values shown on the map were taken from electric logs and/or drill stem tests. On some wells it was impossible to obtain an accurate depth for the water-oil contact due either to insufficient penetration or to lack of data. No values are shown for these wells. Of the wells that penetrated the formation sufficiently, only Imperial Excelsior No. 13 failed to encounter water. The bottom of the Nisku was found at 1,619 feet subsea, which is a few feet higher than the water level in adjacent wells. This fact points to another favour-

Table I.— Basic Data, Excelsior Reservoir

Productive area	1,680 acres
Maximum oil zone thickness	163 feet
Average oil zone thickness	80.5 feet
Connate water (estimated)	20 per cent
Shrinkage factor	0.858
Average porosity	4.93 per cent
Recoverable stock tank oil-in-place	17,743,000 bbl.*
Original reservoir pressure	1,245 p.s.i.g.
Saturation pressure	915 p.s.i.g.
Reservoir temperature	120°F.
Solution G.O.R.— Flash liberation	272 cu.ft./S.T. bbl.
Stock tank oil gravity	35.0°A.P.I.
Approximate number of locations	40

*Assuming 50 per cent flushing efficiency.

*Imperial Oil Limited, Calgary.

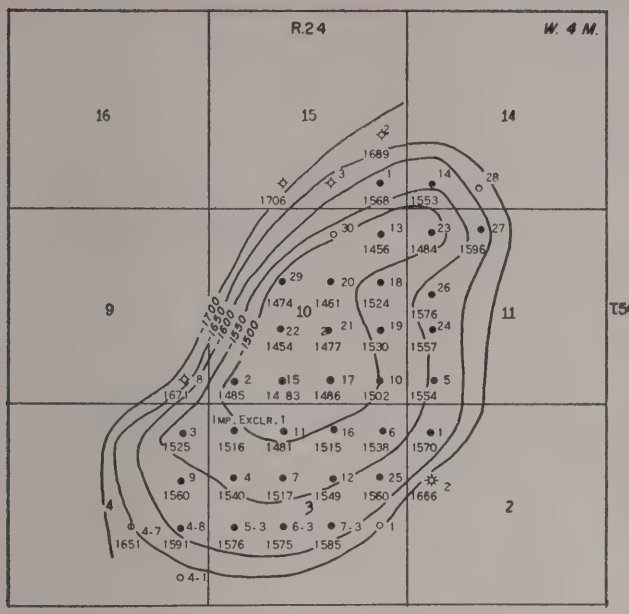

Figure 1.—Contours on top of Nisku.
Contour interval — 50 feet.

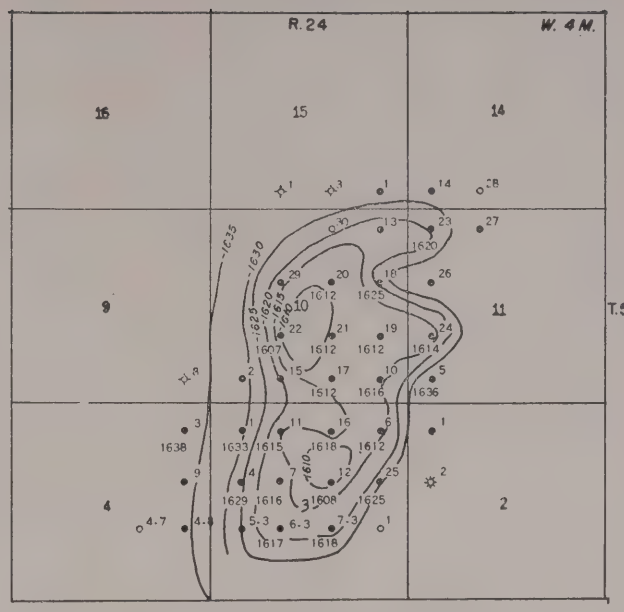

Figure 2.—Contours on water-oil contact.
Contour interval — 5 feet.

able characteristic of the field — since nearly all the oil zone is underlain by water, the mechanism of flood will be essentially a bottom-water drive as compared to the often found, but less efficient, edge-water drive.

Figure 3 is an isopach map of the productive section. The maximum thickness is 163 feet, which decreases to zero as the edge of the pool is approached. There is no gas cap in this reservoir as the crude is undersaturated.

The effective permeability throughout the section is very high. Flowing well tests give a permeability range from 250 to over 8,000 millidarcys. Other tests of the Nisku in surrounding areas indicate that these excellent permeability characteristics extend for many miles in all directions.

Porosity in this reservoir is mainly of the intergranular and vuggy types. An average of 4.93 per cent was obtained from results of analyses of cores from ten wells.

Connate water saturation has been estimated at 20 per cent of pore volume. For intergranular material, connate water may be as high as 50 per cent, and for vuggy material it may be less than 5 per cent. The exact distribution of pore volume between these two types of porosity is not known for this field but a visual examination of cores, and the high permeabilities found on well tests, indicate that the vuggy type predominates and, therefore, that our estimate of connate water saturation is reasonable.

TABLE II.— ANALYSIS OF BOTTOM-HOLE SAMPLE, IMPERIAL EXCELSIOR NO. 1 WELL

Date	February 15, 1950
Sampling depth	1,527 feet subsea
Flowing pressure at sampling depth	1,057 p.s.i.g.
Temperature at sampling depth	120°F.
Saturation pressure	915 p.s.i.g.
Compressibility	7.4 x 10^{-6} bbl/bbl/p.s.i.
Formation volume factor: Flash liberation	1.1 8 resvr. bbl/S.T.bbl.
Differential liberation	1.145 resvr. bbl/S.T.bbl.
Solution gas-oil ratios: Flash liberation	272 cu.ft./S.T.bbl.
Differential liberation	254 cu.ft./S.T.bbl.
Gravity of stock tank oil	35.0°A.P.I.
Viscosity of reservoir oil at 120°F. and 915 p.s.i.g.	1.355 centipoise

RESERVOIR FLUIDS

One bottom-hole sample from Imperial Excelsior No. 1 has been obtained and analyzed. A summary of the data is given in Table II. The oil is undersaturated and has a bubble point pressure of 915 p.s.i.g. at the reservoir temperature of 120°F. When flashed from saturation pressure to atmospheric pressure, the fluid releases 272 cubic feet of gas per barrel of surface oil. The formation volume factor is 1.168 reservoir barrels per stock tank barrel. Liquid volume shrinkage is 14.5 per cent based on reservoir oil and 16.8 per cent based on surface oil. Gravity of the stock tank crude is 35.0°API. Graphs of the formation volume factor for both flash and differential liberation are shown in Figure 4. Compressibility of the reservoir fluid is 7.4 x 10^{-6} bbl/bbl/p.s.i. The viscosity of the reservoir fluid at 120°F. decreases from a value of 1.40 centipoises at the original reservoir pressure to 1.355 centipoises at saturation pressure and then increases rapidly for lower pressures due to liberation of gas from the fluid. A curve of viscosity as a function of pressure is shown in Figure 5.

OIL-IN-PLACE

The total estimated productive area of the field is 1,680 acres. From planimetering the isopach map shown in Figure 3, a total rock volume of 135,000 acre-feet was obtained. This figure and the average porosity of 4.93 per cent gives a total pore volume of 6,663 acre-feet, or 51,695,000 barrels. The original oil-in-place was calculated in the following manner:

Pore volume	6,663 acre-ft. or 51,695,000 bbl.
Connate water satur'n	20 per cent
Hydrocarbon saturation	80 per cent
Hydrocarbon volume	41,360,000 bbl.
Shrinkage factor	0.858.
Stock tank oil-in-place	38,487,000 bbl.

By dividing the rock volume in acre-feet by the total pool area in acres we arrive at an average thickness of 80.5 feet for the field. Each acre-foot of rock volume contains 262 barrels of stock tank oil. Water flooding results indicate a recovery efficiency of approximately 50 per cent, or 131 barrels per acre-foot. The recoverable oil-in-place figures are:

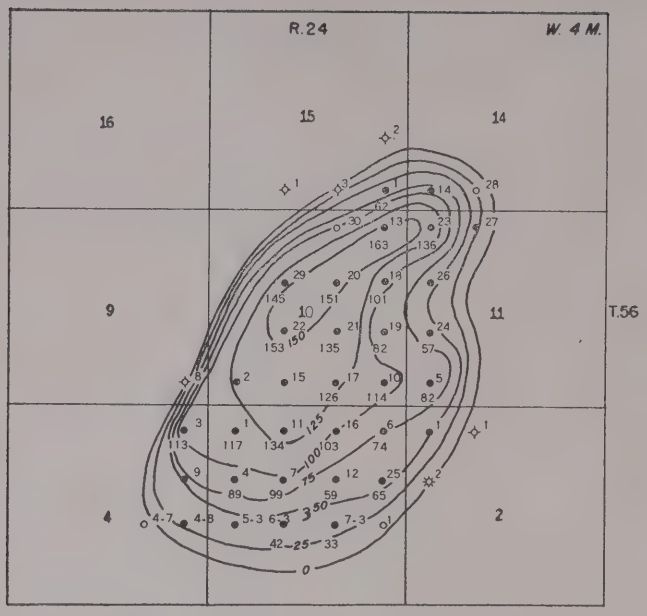

Figure 3.—Productive interval isopach.
Contour interval — 25 feet.

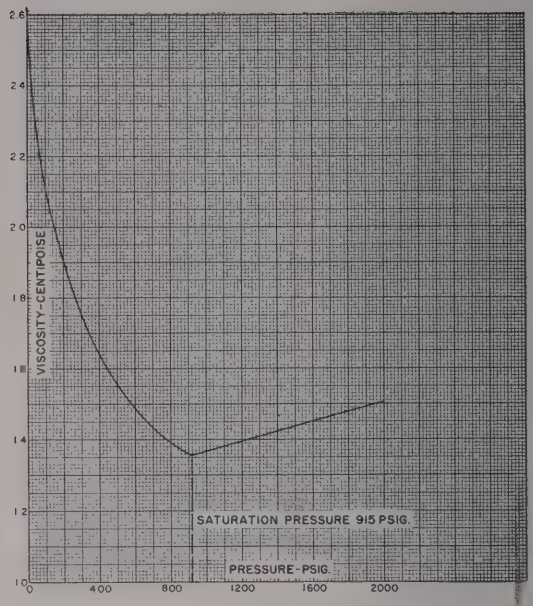

Figure 5.—Excelsior reservoir.
Viscosity of reservoir oil *vs.* pressure.

Recov. oil per acre-ft..	131 S.T.bbl.
Average recov. oil per acre.....	10,545 ST..bbl
Recov. oil per av. well.	421,800 S.T.bbl
Total recov. oil-in-place.....	17,743,000 S.T.bbl

Pressure-Production History

By December 31st, 1951, 997,000 barrels of stock tank oil had been produced. This amounts to 5.6 per cent of the original recoverable oil-in-place. Original reservoir pressure has been estimated at 1,245 p.s.i.g.

The latest survey in mid-December, 1951, gave an average pressure of 1,235 p.s.i.g. for the field, which is a decline of 10 p.s.i. from original conditions. This indicates an average production of 100,000 S.T. barrels of oil per pound drop in pressure since discovery. During the latter half of 1951 this production approached 200,000 S.T. barrels per p.s.i. The maximum variation of bottom-hole pressures during any one survey has been 10 p.s.i. No well has shown a tendency to have the highest pressure for each survey nor has any well consistently exhibited the lowest pressure. Therefore, it is felt that these pressure differences are due only to small statistical errors in the bomb itself, and that no pressure gradient exists across the field.

A summary of pressure-production statistics is given in Table III and is shown graphically in Figure 6. Up to this time, water production has amounted to less than one per cent of the total fluid withdrawals and will not be considered in this paper.

Volumetric Balance

A simplified volumetric balance calculation has been made for this field. The total reservoir withdrawals to the end of 1951 were approximately 1,162,000 barrels and the cumulative water influx at the same date was calculated to be 1,159,000 barrels.

The replacement efficiency of a water drive is expressed as the volume of water influx divided by the total reservoir withdrawals. For Excelsior we have:

Replacement Efficiency
$= 1{,}159/1{,}162 = 99.7\%$

which indicates that this is essentially a complete water drive pool.

'Unsteady State' Calculations

The 'unsteady state' equation, derived by Hurst (1), has been applied to this field in an attempt to predict pressure behaviour for several rates of withdrawal. The equation had a tendency to be unstable in these calculations because of the very small pressure drop since discovery. However, in spite of the instability, it is felt that there has been enough field history to enable us to make reasonably reliable predictions for two years into the future.

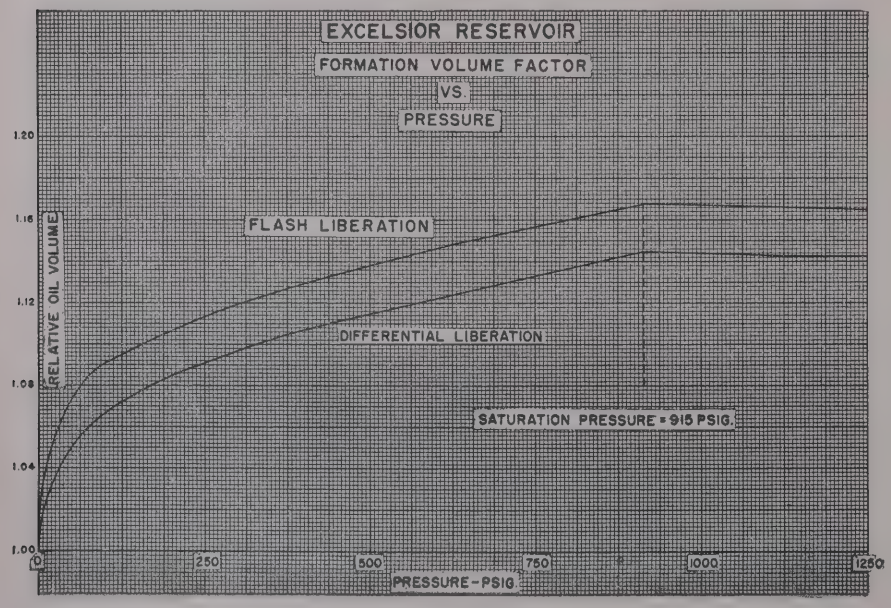

Figure 4.

(1) Hurst, W., A.I.M.E., Trans., Vol. 151, p. 57, 1943.

TABLE III.—PRESSURE-PRODUCTION STATISTICS

DATE	AVERAGE PRODUCTION RATE S.T. bbl/day	MONTHLY PRODUCTION S.T. bbl	CUMULATIVE PRODUCTION S.T. bbl	RESERVOIR PRESSURE p.s.i.g.	SURVEY DATE
Original.....				1,245	
19					
49					
December....	52	1,616	1,616		
1950					
January.....	41	1,257	2,873		
February....	125	3,488	6,361		
March.....	296	9,167	15,528		
April.....	190	5,707	21,235		
May.....	532	16,498	37,733		
June.....	816	24,481	62,214		
July.....	795	24,648	86,862	1,243	July 11
August.....	1,004	31,122	117,984		
September...	1,266	37,975	155,959	1,243	Sept. 26
October.....	1,282	39,733	195,692	1,242	Oct. 9
November...	1,622	48,663	244,355		
December....	950	29,447	273,802	1,242	Dec. 5
1951					
January.....	1,356	42,052	315,854		
February....	1,153	32,275	348,129		
March.....	968	30,023	378,152		
April.....	1,064	31,928	410,080		
May.....	2,200	68,203	478,283		
June.....	2,628	78,861	557,144		
July.....	2,642	81,891	639,035	1,237	July 3
August.....	2,935	90,990	730,025		
September...	2,759	82,783	812,808		
October.....	2,480	76,878	889,686		
November...	2,022	60,664	950,350		
December....	1,501	46,538	996,888	1,235	Dec. 13

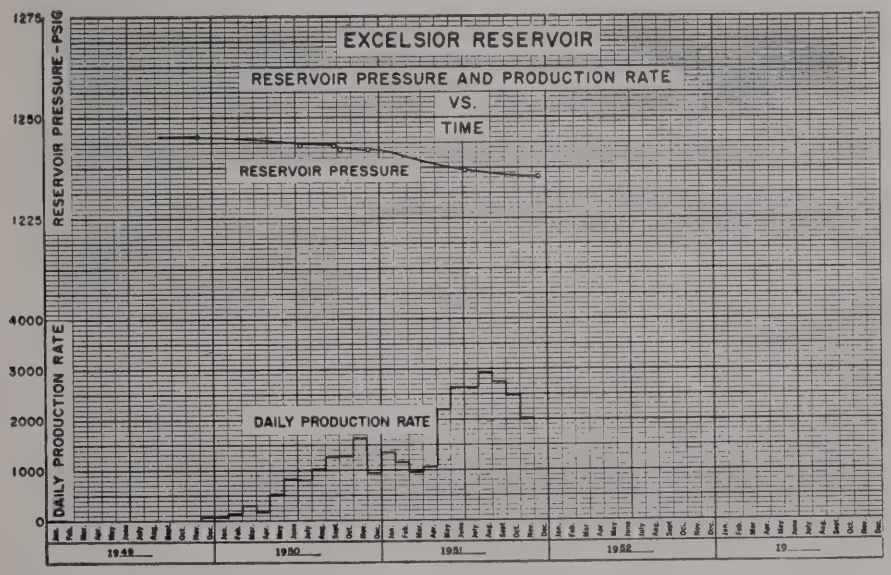

Figure 6.

A plot of the pressure predictions as a function of time is shown in Figure 7. Withdrawal rates of 2,500 bbl/day, 5,000 bbl/day, and 10,000 bbl/day were assumed so as to give a range of possible future pressure histories. When referring to this graph there are two important things that should be noticed. The first of these is the expanded pressure scale. The total pressure drop shown is from 1,245 p.s.i.g. to 1,207 p.s.i.g., which is 38 p.s.i. or only 3 per cent of the possible pressure drop. Secondly, it should be noted that the saturation pressure for this reservoir is 915 p.s.i.g., which is some 300 p.s.i. below the curves.

The curve of daily production rate *versus* time shows that the average withdrawals for the latter half of 1951 were approximately 2,500 bbl/day. The graph of predicted pressures for the assumed rate of 2,500 bbl/day joins with the observed pressure history in a smooth curve. This fact speaks well for the accuracy of the predictions. Another interesting point to note is that, for a rate of 2,500 bbl/day, the pressure will level off at a value of approximately 1,230 p.s.i.g., which is only 15 p.s.i. below the original pressure.

The curve of predicted pressures for a rate of 5,000 bbl/day breaks away from the observed trend and drops several p.s.i. below the curve for 2,500 bbl/day. It has been estimated that this curve will become level at about 1,200 p.s.i.g. or 45 p.s.i. below the original pressure.

The pressure curve for 10,000 bbl/day drops rather quickly at first but it, too, begins to show a tendency to become level after two years.

The production to date has amounted to approximately 100,000 barrels per pound drop in pressure. If we project this rate into the future, the pressure drop necessary to produce the estimated recoverable oil-in-place of 17.7 million barrels will be 177 p.s.i. This indicates that the reservoir pressure at depletion will be approximately 1,068 p.s.i.g., which is still considerably above the saturation pressure of 915 p.s.i.g.

GRAVITY DRAINAGE

Some calculations have been made to ascertain the effects of gravity drainage at various producing rates, using the simplified method proposed by H. J. Welge (2). Results of these calculations are shown graphically in Figure 8. The calculated average horizontal permeability from well tests is about 1,000 millidarcys. Often, in reservoirs of this type, the ratio of horizontal to vertical permeability is as high as 10 to 1. Therefore, a value of 100 millidarcys has been assumed as the vertical permeability. Calculations were also made using a vertical permeability of only 10 millidarcys. The two rates of production used were 5,000 and 10,000 S.T. barrels per day.

The interpretation of these curves is as follows: If a volume of 5,000 S.T. barrels per day was to be drawn from the very crest of the reservoir and the vertical permeability was 100 millidarcys, over 99 per cent of the ultimate oil recovery would be obtained before any water

(2) WELGE, H. J., *A Simplified Method for Computing Oil Recovery by Gas or Water Drive*; Journal of Petroleum Technology, Vol. 4, p. 91, 1952.

was produced. For the case of 5,000 barrels per day and only 10 millidarcys, we would obtain 96 per cent of the ultimate oil recovery before any water was produced. For a rate of 10,000 barrels per day and vertical permeabilities of 100 millidarcys and 10 millidarcys, the ultimate oil recovery before water breakthrough is 99 and 93 per cent, respectively.

Of course, this concept of withdrawing oil only at the very crest of the structure cannot be applied to this field, since production is being taken from all levels of the producing section. These calculations do, however, illustrate the fact that the reduction in oil saturation due to the sweeping action of the invading water is small compared to the reduction in oil saturation brought about by gravity effects.

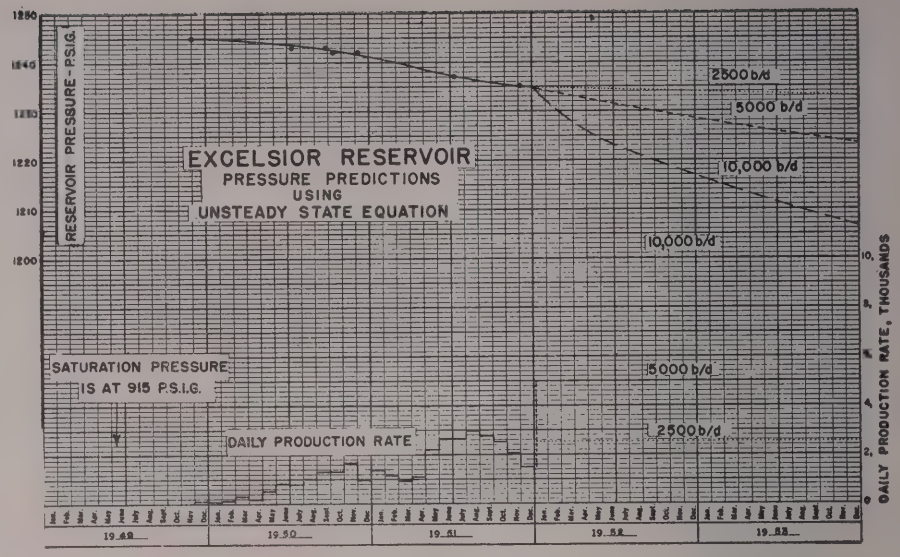

Figure 7.

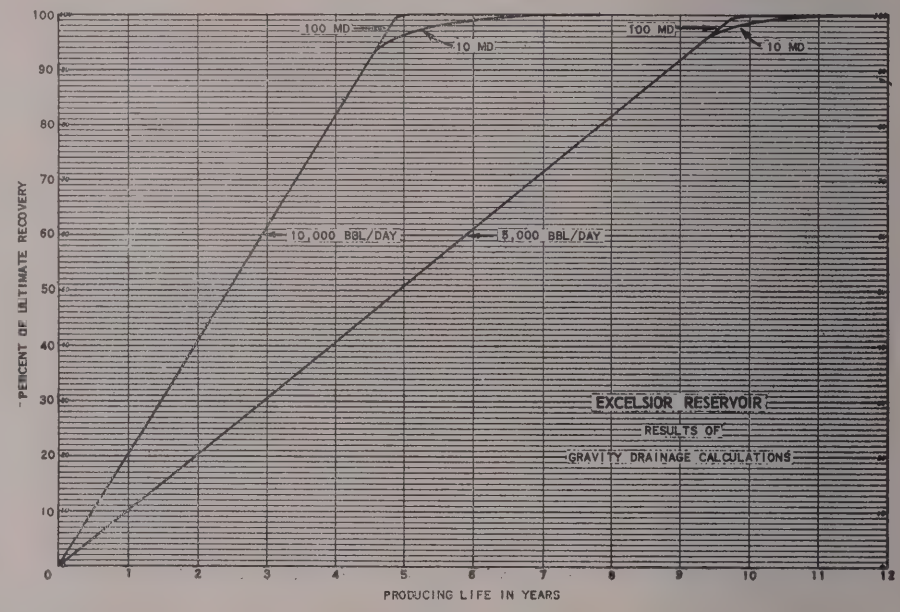

Figure 8.

Maximum Efficient Rate

The maximum efficient rate of oil production from a reservoir is defined as the highest rate that may be maintained for an appreciable period of time without creating substantial underground waste. This rate, commonly abbreviated MER, should not be misconstrued as a 'most efficient' or 'optimum' rate, since these terms infer a single rate of maximum efficiency, and it is believed that no such rate can be determined for a reservoir. Instead, the MER is regarded as the upper limit of a range of efficient rates. Any increase above this limit will cause avoidable waste, whereas any decrease will not materially raise the ultimate oil recovery.

There are two factors to consider when setting an MER for a particular field. These are:

(1) The productive capacity of the field must not be exceeded.

(2) The individual well rates must not be excessive.

During the early part of a field's history, the MER is usually governed by the capabilities of the individual wells. After the field has been completely developed, the MER is governed by the capabilities of the field as a whole. The smaller of these capacities should fix the MER for a reservoir .

Other factors affecting the MER of a reservoir are:

(1) Production rates should not be so large as to cause excessive pressure declines in any one area.

(2) Rates should be low enough to allow the water table to rise evenly over the whole structure. In heterogeneous reservoirs, a rapidly advancing water table will tend to rise faster in the permeable strata and sometimes leave oil trapped in tight sections. A slow rate of advance will allow capillary action to flush these tight sections.

(3) Rates should be low enough to minimize water coning.

It has been found that the size of a field, or its areal extent, has a pronounced effect on the MER for a water drive reservoir. Of two water drive fields with similar characteristics, the smaller can be produced at a higher percentage depletion rate than the larger and still maintain the same recovery efficiency. This is due to the fact that the smaller field has a proportionally greater surface for water influx per unit of oil-in-place.

Conclusions

The MER for any field is a function of several variables. These are:

(1) Type of drive and its strength
(2) Areal extent of the field
(3) Degree of pressure stabilization across the field
(4) Production capacities of individual wells
(5) Water coning

For Excelsior, the volumetric balance calculations show that there is an exceedingly strong water drive which is effecting a 99.7 per cent replacement of all reservoir withdrawals. Pressure predictions made using the 'unsteady state' equation indicate that this water drive will continue strong in the future. For any reasonable production rate, the reservoir will still be above saturation pressure at the time of depletion.

The average permeability of the producing formation is approximate-

ly 1,000 millidarcys. The pressure draw-down for an average well in the field has been calculated to be less than 10 p.s.i. for a production of 200 barrels per day. Gravity drainage calculations show that oil produced by the sweeping action of the water will be small compared to that produced by gravity effects. The water table in Excelsior is rising very slowly and will probably never exceed a velocity of 15 feet per year. The porosity and permeability characteristics seem to be quite uniform, and it is very likely that the water level will rise evenly throughout the structure.

To date, no pressure gradients have appeared across this field and there has been very little evidence of water coning.

All these facts, together with the small areal extent of the field, show that this reservoir is capable of producing efficiently at rates higher than those in the past. Present data indicate that the MER for the early life of this field is probably as high as 8,000 to 10,000 stock tank barrels per day.

Acknowledgment

The writer wishes to express thanks to the Management of Imperial Oil Limited for permission to publish this paper, and to members of the engineering staff who have greatly assisted in its preparation.

CONTRIBUTED DISCUSSION

ON

The Influence of Roasting Temperature Upon Gold Extraction by Cyanidation from Refractory Gold Ores*

By R. CARTER and C. S. SAMIS

Transactions, Volume LV, 1952, pp. 326-327)

MR. L. E. DJINGHEUZIAN (1): The work carried out by the authors is another step forward in the study of the fascinating subject of roasting.

After some preliminary tests on cyanidation and roasting of raw concentrate, the authors made numerous tests on a 'distilled' concentrate, *i.e.*, on a product obtained by volatilizing the arsenic from the raw concentrate.

They have taken several photomicrographs of roasted products. These photomicrographs show grains to which Haszard (1)† and Norwood (2) referred as having a structure similar to 'puffed rice', this structure being typical of 'flash" roasting or particle fusion.

Tests *C-53, -50, -52, -49, -63, -62,* and *-64* show that, at lowest temperatures at which roasting is possible, the introduction of high-sulphur-dioxide atmosphere into the oxygen atmosphere has very little effect on roasting results. This is in line with Norwood's investigations. Norwood found that, by roasting Kalgoorlie concentrates in either air or sulphur dioxide at low temperatures, roasting results were about the same. However, when using higher temperatures, he found that, with roasting in sulphur dioxide atmosphere, the harmful effect of high temperature in roasting in air was greatly reduced. Norwood explained this by the fact that the oxidation of pyrite and pyrrhotite in sulphur dioxide atmosphere was endothermic. Hence the temperature of particles could not rise above that of the furnace and thus very little of particle fusion took place.

In explaining the causes of particle fusion or 'flash' roasting the authors suggest a combination of two mechanisms, namely:

(1) Particle temperature which, due to the highly exothermic reaction causing overheating, is much higher than the furnace temperature; and

(2) A FeO-FeS eutectic formation.

The idea of eutectic melting is a novel one. The eutectic melting is a function of the ratio of reactants and is probably a far more important physico-chemical factor in the mechanism of roasting than has ever been realized. As suggested by the authors, the study of this factor involves the consideration of phase diagrams.

The most striking feature of the work done by the authors is the use of the autoclave for the oxidation of the 'distilled' concentrate, which is another extension in the application of this remarkable apparatus. The potentialities of the autoclave were recognized by Professor F. A. Forward in 1947 in developing his process for treating Lynn Lake concentrates, and since that time its applications in extractive metallurgy have been spreading like a prairie fire.

The results obtained by the authors using the autoclave for oxidation of the 'distilled' concentrate, as shown by tests *C-106, -107,* and *-108,* are of utmost interest: almost a completely porous product was obtained. It is obvious that the formation of the porous ferric oxide was due to reactions taking place under pressure in presence of NaOH and oxygen, and, in oxidizing 200 grams of 'distilled' concentrate, 236 grams in NaOH was consumed.

The use of NaOH suggests that the addition of common salt in roasting of the 'distilled' concentrate might have produced highly beneficial results. At Lake Shore*, without the addition of common salt to the roast, no calcines of sufficient porosity to make the process commercial could be produced. The addition of salt appeared not only to speed up the roast but also to lower the temperature in the initial stages of roasting.

To conclude my remarks, I would like to add that not long ago Mr. A. L. Blomfield, of Lake Shore, suggested that a roaster is not much concerned with the free cyaniding gold in its feed, but only with gold to be freed by the roast. Hence, if B is that part of the head value which is to be freed by the roast and R is the final residue obtained after cyanidation of the calcine, the extraction due to roasting is:

$$\frac{(B - R) \times 100}{B}$$

Since 31.1 per cent of the gold can be extracted by cyanidation of the raw concentrate used in the authors' investigations, by deducting 31.1 per cent of the gold from calculated total head values, values for B for different tests have been compiled and are shown in the accompanying Table:

In a Mines Branch investigation on a concentrate containing 2.77 oz. gold per ton and 26.3 per cent arsenic, the cyanidation of the raw

*See March, 1952, BULLETIN, pp. 160-166; *Trans.*, Vol. LV, 1952, pp. 130-136.

(1) Scientist, Mines, Division of Mineral Dressing and Process Metallurgy, Mines Branch, Ottawa.

†For references, see the authors' paper.

*WILLIAMSON, J. E., *Roasting and Flotation Practice in the Lake Shore Mines Sulphide Treatment Plant;* C.I.M., Trans., Vol. L, 1947, pp. 138.200

concentrate gave 82.7 per cent extraction and the cyanidation of the roasted concentrate 95.8 per cent. The straight cyanidation residue was 0.48 oz. per ton and the cyanided calcine residue 0.17 oz. per ton, equivalent to 0.11 oz. per ton of the raw concentrate. Hence, *B* being equal to 0.48 oz. per ton, the extraction due to roasting was 77 per cent.

CALCULATED EXTRACTIONS OF GOLD FREED BY THE ROAST

TEST NO.	MAX. TEMP., °C.	TIME AT MAX TEMP., hr	B Au. oz./ton	R Au, oz./ton	EXTRACTION %
	450	11	1.11	0.53	52.3
C-53	450	3	1.64	0.48	70.7
C-49	450	7	1.82	0.46	74.7
C-54	450	9	1.69	0.52	69.2
C-71	650	6	1.81	0.10	94.5
C-108	150	1	—	—	97.2

I submit that this example indicates why the accompanying Table, based on Blomfield's formula, gives a more accurate picture of roasting results than do the figures in the authors' paper.

DISCUSSION

ON

A Nomenclature of Igneous Rocks*

By I. C. BROWN

(*Transactions, Volume LV*, 1952, pp. 327-328)

DR. J. W. AMBROSE (Queen's University): I am glad to have this opportunity to comment on the scheme of nomenclature as proposed by Ira Brown. There are two points, among several that could be made, that I wish to mention. Some preamble is necessary to develop my first point. Before rocks can be named intelligently they must be classified—that is to say, a system of nomenclature is, in the final analysis, a statement of a system of classification.

Classifications of rocks may be based on chemical or mineralogical compositions, on textures, on manner of occurrence, or on concepts as to their origin.

Even if rocks were simple chemical systems, which they are not, or if a given rock composition always resulted in development of a given set of minerals, which does not happen, or if inferences as to occurrence and origin could be eliminated, which commonly cannot be done, several systems of classification could still be developed, each with its distinctive features of nomenclature. Obviously, since rocks are variable almost without limit, a simple classification that will satisfy all needs and serve all purposes cannot be devised. As a result, we have two dozen or more different systems of classification, ranging from purely mechanical-chemical, as the C.I.P.W. system, to volumetric-mineralogical, as the Johannsen system. Every system has merits, depending on the point of view or the proposed use; each has its defects, some of them so serious as to make use of the system impractical.

As a result, petrographic nomenclature, which for each author or school is controlled by and expresses viewpoints developed in the favoured classification, is chaotic, to say the least. Granite to one man is a deep-seated intrusive composed of quartz and an alkalic feldspar; to another, granite is a leucrocratic hypidiomorphic aggregate of quartz and potash feldspar with only minor plagioclase; and a third man is prepared to argue that few if any granites are intrusive, and to ask in belligerent tones, "What does deep-seated mean anyway?"

This brings me to my first point: the variability of rocks and of the viewpoints of those who examine them is so great that conflicts in definition, classification, and nomenclature are, at once, inevitable and irreconcilable. And yet we are confronted with a practical problem that must be faced daily, since, for intelligent discussion, rocks must be named. How shall we name them? Uniformity in usage is at once eminently desirable and completely impossible if it is to be reached by general agreement. It can only be established by arbitrary ruling by an authority that can make the ruling stick. The Geological Survey, through its publications, exerts an enormous influence in such matters in this country. If this scheme of nomenclature—which is, by and large, a reasonably satisfactory, simple system—is adopted and rigidly applied by the Survey in its publications, it will, I predict, come fairly rapidly into general use throughout Canada. Otherwise it is just another good idea.

My second point concerns two items of the proposed nomenclature. First, I object vigorously to usage that would make oligoclase a varietal mineral in biotite-oligoclase granite, and albite an essential mineral in biotite albite granite. The word 'oligoclase' is superfluous, and the 'albite' granite with 60 per cent albite and 15 per cent orthoclase should be re-named albite granodiorite and placed in a series—albite granite, albite quartz monzonite, albite granodiorite, and trondjhemite—in all of which albite is an essential mineral. Second, I see no necessity to set up a column for monomineralic rocks. Anorthosite is

* January 1952 BULLETIN, pp. 54-57; *Transactions*, this volume, pp. 53-56.

nothing more nor less than a leucocratic gabbro, a gabbro poor in mafic minerals; pyroxenite, peridotite, and dunite form a series based on relative proportions of pyroxene and olivine. Perknite is superfluous, and as for hornblendite, I have yet to be convinced that it belongs in a table of igneous rocks.

However these and other similar points are perhaps better deferred for leisurely examination with the author.

In sum, I support wholeheartedly the idea of some such system of nomenclature as proposed; I think its only chance of success is to have it adopted and rigorously applied by the Survey in all of its publications; and I can find several points in the table itself for discussions that might range from mild to violent. Thus, while reserving the right to differ with the author concerning detail, I nevertheless wish to compliment him and his associates warmly for their constructive approach to a difficult problem, and to wish them success in securing its general adoption—with suitable modifications, of course.

Dr. Brown: I would like to thank Dr. Ambrose for his discussion and to reply to a few of the points he has raised.

With regard to universal adoption of this nomenclature, such a goal was neither intended nor expected. It was hoped that the majority of workers, as has happened in the Geological Survey, would reconcile their differences in the interests of clarity and simplicity. This nomenclature will not be forced on anyone, including members of the Geological Survey, but it is hoped that, in all reports, it will be clearly stated whether this or some other classification is used.

I agree with Dr. Ambrose that 'oligoclase' as a varietal mineral in granite is normally superfluous, as only diagnostic minerals are intended to be used as prefixes. 'Albite granite' is a term about which there was much discussion and it was used rather than 'soda granite' or 'albite granodiorite' because it seemed to be more firmly established in the literature. Less common rock names such as 'trondjhemite' are not included in the table for the sake of simplicity, but are not discarded. They may be used provided they are indicated as varieties of rocks shown in the table.

The monomineralic rocks are not set in a separate column with the intention of calling them a series nor of separating them from their close relatives. The rock names appearing in the nomenclature table are in common use in Canada and for this reason it was decided to show them separately rather than include them with other rock groups. The name 'perknite' was included because the majority of persons consulted favoured its inclusion, in view of the fact that the term 'peridotite' does not include rocks with less than 5 per cent olivine.

The writer would like to emphasize that this nomenclature is not necessarily in its final form. If it appears that the majority of workers favour certain changes, these will be made.

CONTRIBUTED DISCUSSION

ON

The Rod Mill in the Sullivan Flow

By H. R. BANKS

(*Transactions, Volume LV*, 1952, pp. 339-342)

NOTE.—Mr. Banks' paper, which was presented at the Ottawa General Meeting, January, 1952, was published in the July BULLETIN, pp. 397-403, and is included in this volume of the *Transactions*, pp. 255-261.

Appended to the paper was 'contributed discussion' by Mr. B. S. Crocker, Lake Shore Mines, Limited. In connection with Mr. Crocker's comments, we wish to call attention to a misprint on line 13, centre column, BULLETIN page 403, *Transactions* page 261, where it is stated that the mill is doing very coarse grinding "with a feed 2 per cent plus-1-inch". This should read "with a feed 21 per cent plus-1-inch".

The following additional discussion has been received since publication of the paper in July.

MR. L. T. HIGGINS (Mill Superintendent, Braden Copper Company, Sewell, Chili): I would like to congratulate Mr. Banks on his paper about the Sullivan rod mill. To those who have seen this mill in operation, it presents a remarkable performance — grinding about 8,000 tons per day to the 8-10 mesh size. The high lifter bars are an innovation; but I am wondering why such high bars are necessary, and how much extra power they use. Other operators are satisfied with low lifter bars, and some use only smooth liners.

From 1928 to 1936, I operated several 8 ft. by 12 ft. rod mills at various speeds between 12.8 and 22.6 r.p.m. There was considerable pounding at the higher speeds; this is hard on liners and causes rods to break up at a larger diameter than necessary. My conclusion was that high speeds may produce more tons of finished product, but that intermediate speeds give a better operating condition and are more economical.

I am sure that we shall see more installations of the large rod mills, and that fine crushing will become less important.

MR. HARLOWE HARDINGE (Hardinge Company, Inc.): It is stated in the paper that the feed is 21.3 per cent plus 1-inch. I understand that the amount remaining on 2-inch is approximately 3 per cent. Is this correct?

MR. BANKS: Yes.

MR. HARDINGE: I understand that the rod lengths in the mill are 11 ft. 6 in., but that with this type of convex head mill a full 12-foot rod can be used without causing the rod ends to wedge against the end lining.

MR. C. G. MCLACHLAN (General Superintendent of Mills, Noranda Mines, Limited): I was extremely interested in Mr. Banks' paper, particularly as my first association with rod milling goes back over twenty-five years. At that time two-stage rod milling was being employed at the Nacozari plant of the Phelps Dodge Corporation in Mexico, and rod milling was also being carried out at the Copper Queen mill in Bisbee, Arizona. The primary mills at both these plants were taking a relatively coarse feed and handling it without difficulty. Their performance from the standpoint of metallurgical recovery was also entirely satisfactory. Then in 1928, and 1929, high metal prices made it desirable to try to obtain more production. I believe it was felt that the rod mills were then running at their maximum speed, but that if they were converted to ball mills they could be run faster and more power could be put into them and, therefore, they would grind more ore. Consequently the rods were taken out and replaced with balls. It would now seem that, with the conception of the high-speed rod mill, the development cycle of this machine has been completed. I would like, however, to offer a word of warning. I think there is no question now that the rod mill has demonstrated its usefulness and adaptability. Moreover, it is my personal opinion that its best application is for treating relatively large tonnages, *i.e.*, 800 tons or more a day of ore which is too coarse to be treated advantageously by a ball mill. I would like, therefore, to suggest that when possible applications of rod mills are being considered, they be engineered on their individual merits and that care be taken to ensure that their installation is not based on foregone conclusions.

* * *

Interpretation of the Sullivan Rod Mill in Comminution

By J. F. MYERS, Tennessee Copper Company, Copperhill, Tenn.

To this writer's belief, a new fundamental contribution to crushing with rod mills has been made by H. R. Banks of the Consolidated Mining and Smelting Company, at Chapman Camp, British Columbia, and in a minor way by B. S. Crocker of Lake Shore Mines, at Kirkland Lake, Ontario. In both of these mills the length and diameter are not far different, their dimensions being 11½ ft. by 12 ft. and 7 ft. by 6 ft., respectively. Both mills are followed by several stages of ball mill grinding, three at Chapman Camp and four at Lake Shore. The sole pur-

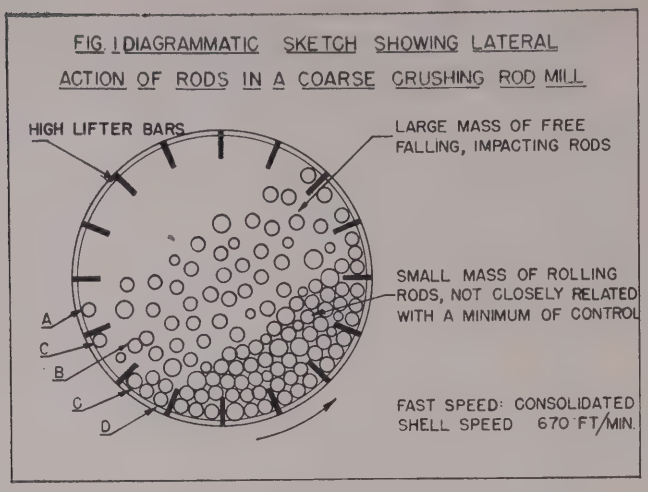
FIG. 1 DIAGRAMMATIC SKETCH SHOWING LATERAL ACTION OF RODS IN A COARSE CRUSHING ROD MILL
HIGH LIFTER BARS
LARGE MASS OF FREE FALLING, IMPACTING RODS
SMALL MASS OF ROLLING RODS, NOT CLOSELY RELATED WITH A MINIMUM OF CONTROL
FAST SPEED: CONSOLIDATED SHELL SPEED 670 FT/MIN.
A
C
B
C
D

Figure 1.

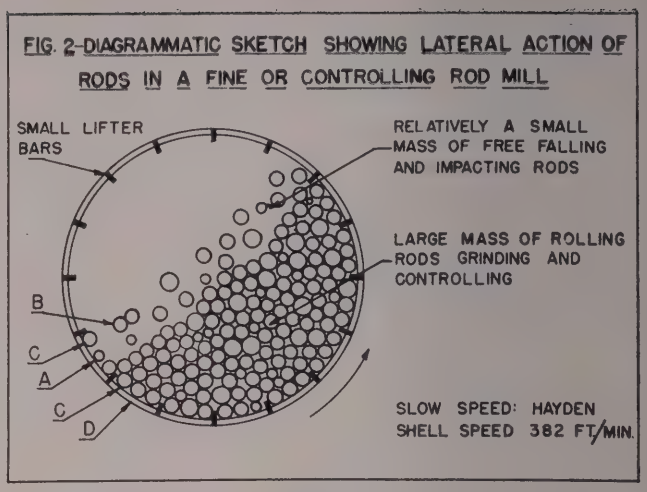
FIG. 2-DIAGRAMMATIC SKETCH SHOWING LATERAL ACTION OF RODS IN A FINE OR CONTROLLING ROD MILL
SMALL LIFTER BARS
RELATIVELY A SMALL MASS OF FREE FALLING AND IMPACTING RODS
LARGE MASS OF ROLLING RODS GRINDING AND CONTROLLING
SLOW SPEED: HAYDEN SHELL SPEED 382 FT/MIN.
B
C
A
C
D

Figure 2.

pose of the Sullivan and the Lake Shore mills is to obtain cheaply a degree of comminution for multiple stages of ball milling that follow them.

In the fall of 1950, the writer visited the Sullivan operation. Through the courtesy of Mr. Banks, he was permitted to vary many of the rod mill adjustments. The very high speed, the big lifter bars — which create a tremendous crushing force as they hit the toe of the rod mass — the high rate of feed, and the entirely unrelated criss-cross mass of rods at the toe, presented an amazing spectacle. It was fascinating to watch the big lifter bars scissor the rods back into position for the return trip up the ascending side.

Fortunately, the Sullivan rod mill has a forty-inch discharge opening and, by means of a powerful floodlight, the entire action within the mill is visible to the feed opening.

Upon leaving Chapman Camp, the writer set forth immediately for Hayden, Arizona, to witness again the quiet, smooth precision action of the Hayden mill, which produces very efficiently an enormous amount of ore surface. How, then, can we reconcile the rod action of the Hayden type rod mill with that of the Sullivan rod mill?

The writer concludes that rod mills, like every other piece of equipment in ore treatment processes, especially crushing, must have different adjustments and dimensions for 'coarse crushing' as distinguished from fine or 'controlled crushing' work, with a compromise arrangement to cover extreme ranges of reduction for small plants as typified by the Tennessee mill.

In Figures 1 and 2 the writer has attempted to show diagrammatically the difference in the lateral rod action of a coarse crushing rod mill and that of a fine or 'controlling' rod mill, based upon his studies and observations. While it might be contended that this is only one man's viewpoint, nevertheless, the observations have been made in the prseence of other engineers.

It will be noted that, in the case of the coarse crushing mill, Figure 1, the fast shell speed (Sullivan rod mill peripheral speed 670 feet per minute) and the high lifter bars have minimized the compact area of rolling rods and amplified the number of free falling rods, causing impacting in the toe area.

Add to this the intense slugging action of the high lifter bars as they enter the toe area and it is obvious that the mill has a tremendous amount of impact crushing capacity, which is essential for the comminution of coarse particles of ore.

In Figure 2, the slow shell speed (Hayden rod mill peripheral speed 382 feet per minute) and normal lifter bars intensify the grinding action and controlling action of the rolling rods and minimize the number of free falling rods for impact purposes. Consequently, the mill has an enormous capacity for producing a relatively fine and controlled product, but little capacity for much coarse crushing or impact work.

Let us now consider the longitudinal rod action as diagrammatically shown in Figure 3.

That the rod mass on the feed end of the mill is swelled up by coarse ore between the rods is easily demonstrated by pulling the motor switch with a fully loaded rod mill. It can be observed readily that the pool level is below the discharge opening, and that the crushed or ground ore leaves the rod mass from the voids between the rods at the discharge end of the 'rod slot'. This has previously been described by

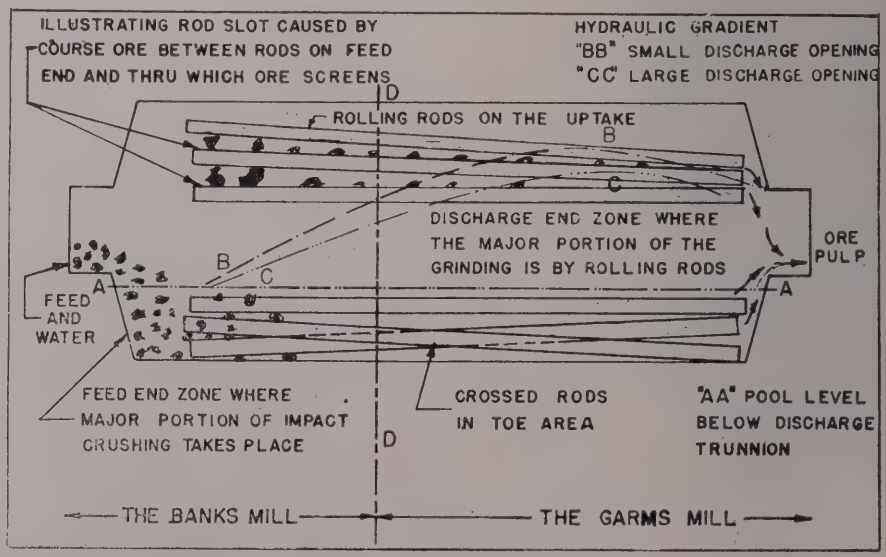
ILLUSTRATING ROD SLOT CAUSED BY COURSE ORE BETWEEN RODS ON FEED END AND THRU WHICH ORE SCREENS
HYDRAULIC GRADIENT
"BB" SMALL DISCHARGE OPENING
"CC" LARGE DISCHARGE OPENING
D
ROLLING RODS ON THE UPTAKE
B
C
DISCHARGE END ZONE WHERE THE MAJOR PORTION OF THE GRINDING IS BY ROLLING RODS
ORE PULP
B
C
A
A
FEED AND WATER
FEED END ZONE WHERE MAJOR PORTION OF IMPACT CRUSHING TAKES PLACE
CROSSED RODS IN TOE AREA
"AA" POOL LEVEL BELOW DISCHARGE TRUNNION
D
THE BANKS MILL
THE GARMS MILL

Figure 3.—Diagrammatic sketch showing longitudinal action of rods, exaggerated for illustration purposes.

Myers and Lewis and is as illustrated in Figure 3.

The crossed or 'jack strawed' rods effect in the toe area has previously been discussed.

To the writer, the crux of the rod mill action is the 'rod slot'. This has not appealed to most investigators. The rod slot is the interstitial space between the rods in the uptake zone and its dimensions are controlled by the ratio of the largest pieces of ore on the feed end to the largest particles in the discharge product on the discharge end.

Obviously, on the feed end, the rod slot serves almost entirely as a means of elevating the coarse ore particles to cascade them to the toe area for impacting and scissor crushing until they become small enough to progress longitudinally toward the discharge end by the screening effect of the rod slots. Fines in the feed quickly progress toward the discharge end *via* the screening action of the rod slots, while the coarse ore pieces are retained at the feed end until adequately reduced in size.

Let us consider the rod action to the left of vertical line *D-D*. In this zone practically all of the work is accomplished by impacting and scissoring. This is the coarse crushing end of the mill. If the mill were cut off at line *D-D* and all factors adjusted to best suit impact crushing and scissoring, we have, in this writer's opinion, the Sullivan coarse crushing rod mill which produces the lateral action shown in Figure 1.

To the right of line *D-D*, practically all of the work is accomplished by the rolling action of the rods slipping and grinding the ore to a controlled product. If the mill were cut off at line *D-D* and all factors adjusted to best suit the rolling action, we have, in this writer's opinion, the fine crushing and controlling Hayden mill as designed by Garms.

The essential differences between these two rod mill installations are summarized in Table I.

The composite mill shown in Figure 3 is the conventional mill so generally found in operation today. It is obviously impossible to think that the composite or conventional mill will ever be as efficient as if the individual actions are separated, as in the Sullivan and Hayden mills, and the two run in series. One might as well argue that a single crusher can efficiently crush rock from, say, 48-inch down to ¼-inch in one machine. This, we know, is not true, and each state of crushing requires both different design and different adjustments.

TABLE I.— COMPARISON OF SULLIVAN AND HAYDEN ROD MILLS

	SULLIVAN MILL	HAYDEN MILL
Ratio of diam. to length	1.047 to 1	1.33 to 1
Shell speed	670 ft./min	382 ft./min
Lifter bars above plates	3¼ in.	1¾ in.
Size of rods	3½ in.	3 in.
Size of feed	1½ in.	⅜ in.
Size of product	−4 mesh	−10 mesh
Size of discharge opening	40 in.	22 in.
End liners	20°	4°

In the tabulated comparison of the Sullivan and Hayden mills it will be noted that the one at Sullivan takes care of the 1½-inch feed produced by closed circuit dry crushing whereas the Hayden mill handles only ⅜-inch feed produced by dry close-circuit crushing. It is obvious why more impact crushing is needed in the case of the Sullivan. For the small plant, open circuit, one dimensional crushing is most economical, as typified by such installations as that at Tennessee.

There exist no metallurgical data that the writer can find which justified the manufacturers to switch from vertical end liners to conical end liners a number of years ago. The various degrees of coning the ends in modern mills is an arbitrary variable adopted by different manufacturers entirely for manufacturing purposes. There exist no data to substantiate the various degrees of coning in the mills manufactured today. At the instigation of the writer, the Allis-Chalmers Company went back over many years of records and made a comprehensive study of vertical *vs* coned end liners. They presented the statement in an advertisement in the January, 1951, issue of the *Engineering and Mining Journal* that vertical end liners were 2 per cent more efficient than coned end liners.

For fine crushing or controlling rod mills handling nominal tonnages, there is no doubt that the vertical end liners on the discharge end seal off to some degree the escaping pulp from the voids between the rod ends, thus forcing the pulp to maintain a high pulp gradient in the rod mass and bringing more screening slots into action, all of which makes for higher efficiency.

Since the major portion of the work performed by the Sullivan mill is impacting, exacting control is a minor point. This is fortunate, as it now appears that steep coned ends will be vital to the coarse crushing mill in order to expedite the removal of large broken rods. In the Sullivan mill, the writer observed 1¾ in. to 2 in. rods breaking into pieces as long as 4 feet. The steep coned discharge end made it possible to shed these broken pieces, which obviously a vertical end could not do.

Much has been written about rod mill shell liners for conventional mills. Space does not permit a review of this subject. However, the very high liners (3¼ in.) in the fast Sullivan mill should be mentioned. As designed by Banks, they gave a 'Joe Louis' punch to the rods in the toe area. At the moment, it would seem that this type of liner should be a part of any coarse crushing rod mill.

* * *

MR. L. J. ERCK (Chief Metallurgist, The Cleveland-Cliffs Iron Company, Ishpeming, Mich.): The author had the pleasure of reading the paper by Mr. Myers prior to its presentation regarding the virtue of the Sullivan Rod Mill and its potential application to the problems of the Tennessee Copper Company.

There are several questions that come to mind which will be discussed briefly. First of these is the similarity between the working function of the Sullivan rod mill and that of the Weston Aero-Fall mill. In order to compare the operations, the Weston paper presented at Columbus on September 26th, 1949 was re-read and the various points noted. Due to the high peripheral speed together with the high lifter bars used, both machines exact a severe blow impact as well as considerable free fall impact.

The degree of difference in preparation of the material being reduced by attrition and abrasion may vary somewhat. It is realized that the Sullivan mill, using rods, varies from the Weston mill, where autogenous grinding media is used. Weston notes in his paper, however, that at times it may be necessary to use an additional charge of steel balls. From this, then, it can be assumed that the principle of grinding is comparable, even though in one case it is applied to a rod mill and in the other to a pseudo ball mill.

The second thought is brought out by reading Myers paper, in wondering whether or not the extreme limits of application have been reached by using the principles of the Sullivan rod mill. It is noted on page 5 of the paper just presented that the rods are 3½-in. in diameter for a feed of 1½-in. top size. The rod mill so employed definitely fills a gap between the last stage of crushing and the first stage of grinding. There has never been a complete compatibility between these two sections of a mill inasmuch as there is always a tendency to establish a compromise which will favour either the crushing or the grinding section. The application of this preliminary stage of grinding as advocated by Banks tends to fill this gap.

Considering this condition, then, as presented in the paper, the question arises in the writer's mind as to whether or not it would be feasible to consider larger rods and coarser feed to this type of rod milling practice.

Mr. L. E. Djingheuzian (Scientist, Mines, Division of Mineral Dressing and Process Metallurgy, Mines Branch, Ottawa): It was with absorbing interest that I studied H. R. Banks' paper. This remarkable paper marks a significant stage in the development of not only the art but also the science of comminution.

The discussion on the paper at the last annual meeting of the C.I.M. in Ottawa, while exceptionally interesting, especially the masterly analysis of the action of the Sullivan rod mill by J. F. Myers, nevertheless hardly touched on what, to me, is probably the most important statement in the paper, namely that:

> *"Comminution begins with the start of the first drill hole in the face of the drift or stope where the ore is mined".*

The data on the operation of the Sullivan rod mill scientifically represent only one of the numerous stages in comminution, whereas the concept that comminution begins with mining is basic to the whole science of comminution (2). The author must be congratulated for having made a notable contribution to the science of comminution by bringing out in a precise manner the implications of this fundamental concept.

What are the practical implications of this concept? As the author very modestly says:

> "There is some possibility, however, that the recognition of differences in fragmentation brought about by new techniques in drilling and blasting may lead at least to a considerable reduction in the duty of the initial crushing equipment".

With reference to F. C. Bond's crushing studies, the author says:

> "This type of basic study should eventually lead us into a more efficient application of the available energy in our problem of comminution".

I would like to suggest that the application of this statement should be extended to the study of drilling and blasting as the first stages in the comminution problem.

At the mines we are always worried about the costs. Do not the author's statements introduce new possibilities of cutting down the costs? In the final analysis, co-ordinated research by the miner and the mill man, based on the author's statements, would appear to lead to the following basic ideal:

Cost of drilling and blasting + Cost of crushing and grinding → Minimum.

A matter that interests me very much is the efficiency of the Sullivan rod mill. I am wondering whether an attempt has been made to evaluate this efficiency?

Recently, for evaluation of comparative efficiencies of comminution machines, F. C. Bond (3) introduced the concept of work index, derived from his new theory of comminution.

Taking the rate of daily tonnage of the Sullivan rod mill at 9,375 tons at 19 r.p.m. and power consumption at 1,000 horsepower, kw.hr. consumed per ton is 1.91. Eighty per cent passing for the feed and the discharge has been roughly obtained from log-log plots of the data in the paper, F = 27,500 microns and P = 2,510 microns. The work index calculated from these figures comes to 13.72 kw. hr./ton.

Carrying this step a little farther, I have made an attempt to calculate the mechanical efficiency which, as defined by Bond (3), is the ratio of useful energy input to the energy supplied to the motor.

Assuming the specific gravity of the rod mill feed at 4.5 and calculating the increase in surface to be equal roughly to 919 sq. cm. per cub. cm., the energy supplied to the motor to obtain this increase in surface of one cub. cm. is 34.11 joules, or 371,182 ergs/sq. cm.

Schellinger's (4) calculations from his recent experiments indicated that the theoretical surface energy of pyrite is of the order of 60,000 ergs per sq. cm. Using this figure — which, I am inclined to believe, is a rather low one and hence conservative — for the Sullivan rod mill discharge, the mechanical efficiency of the rod mill works out at 16.17 per cent. The thermodynamic efficiency, naturally, will be higher than this figure.

The above efficiency figures are purely tentative and are offered with a hope that they might be found sufficiently interesting to bring about more research in this exceedingly important field.

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(2) Bond, F. C., *The Third Theory of Comminution;* A.I.M.E., Mining Engineering, May, 1952; A.I.M.E., Trans., 1952.

(3) *Op. cit.*

(4) Schellinger, A. K., *Solid Surface Energy and Colorimetric Determinations of Surface-Energy Relationships for some Common Minerals;* A.I.M.E., Mining Engineering, April, 1952; A.I.M.E., Trans., 1952.

CONTRIBUTED DISCUSSION

ON

The History of Basic Steel Manufacture at Sydney, Nova Scotia*

By M. R. CAMPBELL

(*Transactions, Volume LV*, 1952, pp. 333-334)

MR. A. JACKSON (Deputy General Works Manager, Appleby-Frodingham Steel Company, Scunthorpe, Lincolnshire, England): I read with interest the paper on the history of basic steel manufacture at Sydney, Nova Scotia, by M. R. Campbell. In view of his reference to Appleby-Frodingham and his visit here some three years ago, I felt impelled to write and give some information on the steel making practice used by this Company. If my memory of the interest and enthusiasm of Mr. Campbell, when visiting Appleby-Frodingham, is correct he will be pleased to have this information.

In the course of his paper he describes a process which "was in use at the Frodingham Works, Scunthorpe". In fact, the same process is still in use and, for comparative purposes, the results taken from one melting shop during the past 12 months are set forth in the accompanying tabulation.

The figures show the considerable variation in iron quality and the degree of activity of the mixer. They also illustrate the removal of important quantities of silicon in the mixer with, at the same time, low iron loss in the slag, which in any case is recovered in the blast furnace. No soda ash is used for desulphurization.

The skull loss between mixers and steel furnaces is low simply because the metal is kept at a sufficiently high temperature to minimize ladle skulls.

No fluorspar whatever is used in the steel furnaces and the slag is all ground and sold as fertilizer, a considerable aid to the costs of steelmaking. Furthermore, its iron content is low compared with flush slags, which often contain up to 25

*June 1952 BULLETIN, pp. 331-339; *Transactions*, Vol. LV, 1952, pp. 217-225)

MIXER

AVERAGE IRON TO MIXER.....1.15% Si 0.100% S 1.02% Mn 1.52% P

RANGE OF SILICON, %	% IRON by WEIGHT	RANGE OF SULPHUR, %	% IRON BY WEIGHT
Under 0.5	0.5	Under 0.05	8.7
0.51—0.70	4.0	0.051—0.070	19.9
0.71—0.90	14.1	0.071—0.090	23.9
0.91—1.10	27.8	0.091—0.120	23.1
1.11—1.30	20.9	0.121—0.175	19.4
Above 1.30	32.7	Above 0.175	5.0

ADDITIONS TO MIXER:
- Oxides.....0.96 cwt/ton
- Limestone.....1.23 cwt/ton
- Steel scrap.....1.66 cwt/ton

SLAG FROM MIXER: 1.81 cwt/ton

ANALYSIS:

SiO_2	26.99 per cent	Al_2O_3	2.86 per cent
CaO	40.5 "	MnO	8.97 "
MgO	3.63 "	P_2O_5	6.87 "
S	0.211 "	Fe	7.8 "
TiO_2	1.29 "		

STEEL FURNACES

ANALYSIS OF IRON TO STEEL FURNACES:

Si	0.27 per cent	S	0.084 per cent
Mn	0.47 "	P	1.20 "

ADDITIONS TO STEEL FURNACES:

Oxides	2.05 cwt/ton	Limestone	1.66 cwt/ton
Scrap	3.64 "	Lime	0.23 "

PRODUCTION OF PIT SCRAP:
- Ingot ends
- Runner legs
- Ladle skull
- Spillage
- Recovery ex. slag

} 0.96 cwt/ton

SLAG FROM FURNACES (all sold) 2.81 cwt/ton

ANALYSIS:

SiO_2	9.8 per cent	Fe	8.8 per cent
P_2O_5	16.78 "	Solubility	91.0 "
CaO	49.0 "		

TAPPING SLAG ANALYSIS (left in furnace):

SiO_2	7.8 per cent	Fe	13.9 per cent
P_2O_5	12.24 "	CaO	49.9 "

FUEL CONSUMPTION.....36.1 therms/ton

REFRACTORY CONSUMPTION (2 years' average): 32.5 lb./ton

DOLOMITE CONSUMPTION.....0.52 lb./ton

AVERAGE PRODUCTION PER WORKING WEEK PER FURNACE.....2,718 tons

per cent of iron in addition to quantities of metallic iron thrown over the furnace doors by the violent reactions which often take place.

Published information frequently gives undue prominence to output and so aids the impression that this is the final criterion of steelmaking efficiency. This is, of course, fallacious as the fundamental basis must be cost, and costs are very considerably influenced by yields. A surprising increase in output is necessary to compensate for the loss of 1 per cent yield. In the same context, one frequently sees records of the merits of oxygen lancing, but how often do these show the cost of the loss of yield which arises due to direct burning of expensive pig iron instead of indirect reduction of ore in the furnace. A critical cost survey will not infrequently indicate that it is cheaper to add extra steelmaking capacity and use the normal process.

Similarly, quite a lot of extra output is necessary to compensate for the loss of saleable fertilizer slag in plants which must of necessity use iron initially high in phosphorus.

In connection with the plant described by Mr. Campbell one may well ask, "Why go to the capital cost of tilting furnaces and then work a process which is identical with that normally conducted in fixed?" The essential merit of tilting furnaces is their ability to remove almost completely and cleanly free running slags as and when necessary.

No-one can logically deny the value that must arise from the use of the clean tapping slag to assist the initial refining of the next charge, and this slag must be especially valuable when dead-soft steels, with their consequent high-iron slags, are being produced.

In fact, the Appleby-Frodingham process is somewhat similar to several of the processes mentioned in the paper in that molten iron is poured into a molten slag relatively rich in oxides and lime, and when this is fully exhausted, low in iron, and high in acids, it can be removed freely from the furnace. Spar additions are quite unnecessary in either the primary or the secondary slag.

As regards history, the first tilting furnace to be built at Appleby-Frodingham was in 1900/1902 and it was scrapped about five years ago. It had a capacity of about 140 tons normally tapping 2 60-ton ladles. Since the turn of the century, Appleby-Frodingham have continually developed the tilting furnace process and now make almost 1,000,000 tons of ingots per year in this manner. It must be admitted that we have had more experience in this practice than anyone elsewhere in the world and, whilst we have not tried all the modifications mentioned by Mr. Campbell as having been introduced during the past fifty years, we have worked from 100 per cent metal to high scrap charges. We know the relative value of flushing slags and the effects of mixers on steelmaking practice.

In view of this knowledge, our firm opinion is that one mixer to three or four 300/350-ton tilting furnaces is the ideal at which to aim, and we also know that, with units of this type we make more steel per annum and of a higher quality than if the mixer was converted to a steel furnace and direct metal used in all of them.

We have repeatedly proved the value of the mixer to steel furnace output when it is off for its repair every 15 months, whilst its effect on uniformity of steel quality has been amply demonstrated many times.

We also know that all kinds of steel can be made in tilting furnaces as the process is, with the exception of the electric, the most adaptable in existence.

In 1947, we brought into operation the beginnings of a new melting shop which at present has two tilting furnaces and one mixer, the first portion to be built. Annual productions were as follows:

1949	.....	227,107 tons
1950	.....	240,736 tons
1951	.....	229,112 tons

It is recognized that these furnaces are much bigger than those shown in the paper by Mr. Campbell, but the bottom consists of 12½ inches of magnesite brick on which is rammed 10 inches of dolomite when the furnace is new. Judging from his paper, the 100-ton furnaces with a lining thickness in proportion could possibly hold upwards of 175 tons, which alone would lead to much greater output.

A longer and very interesting discussion could take place if time were available, but I felt impelled to write this brief note to our good friends in Canada.

I hope they will find the information useful and again try the Appleby process, bearing in mind that it required very many years to fully develop and train personnel to take advantage of its considerable flexibility. The introduction of slag control some ten to twelve years ago was a further contribution to its success.

The process cannot be copied or reproduced without the full understanding and use of its inherent flexibility. Our experience has shown that it requires months or even years of practice because it differs so greatly from fixed furnace practice, so familiar to most steel makers.

Many years ago we had a mixed fixed furnace and tilting shop. On the fixed furnaces were some most excellent melters of whose ability we were very proud. When, however, some few of these were transferred to the tilting furnaces their success was very limited and outputs fell very considerably because they tried to work on fixed furnace principles. A few of the men ultimately acquired the requisite experience, but most of the older hands became only a shadow of their former selves when transferred from fixed to tilting furnaces.

I would like to say how interested I was in reading this paper, which well illustrates the difficulties inherent in producing steel from phosphoric iron which is irregular in composition in other respects.

At Appleby-Frodingham we have the greatest sympathies with the many difficulties they meet in Sydney because they coincide very closely with our own. Our process is one derived from long experience and steady development and we have no doubt that it is better than the flush slag process. On the other hand, I would hesitate to be dogmatic because, as I said at the beginning, the essential criterion is cost, and if the added output, due to flush slag practice, more than compensates for losses in yield and slag value, by increased output, there is little doubt that it is the process to be used under the circumstances prevailing.

DISCUSSION

ON

Pilot Plant Investigations on Cold Water Separation of Bitumen from Alberta Bituminous Sands*

By L. E. DJINGHEUZIAN

(*Transactions, Volume LV,* 1952, pp. 335-336)

MR. DJINGHEUZIAN: I was very pleased to have my paper printed in the January BULLETIN, and so, hoping that most of you have read the paper, all I have to do is to introduce it and say a few words about its highlights.

This is an ore dressing paper. Some of you who have not read the paper might ask: What has ore dressing to do with the oil separation from bituminous sand? A lot, I would say, because ore dressing is an application of fundamentals to the benefication of any useful mineral — ad this icludes oil — that might be found in the earth's crust. In fact, ore dressing is one of the cornerstones on which our civilization rests. The cold-water separation of bitumen is nothing else but an application of the fundamental concept of gravity to oil separation with the use of suitable ore dressing machinery, a machinery which, because of its proven robustness and simplicity, might be just the thing for handling continuously for 365 days, *i.e.*, all year round, such rough material as Alberta bituminous sand. I wonder if I am presuming too much when I say that the cold-water process marks a new milestone in oil technology since it demonstrates the applicability of ore dressing machinery to the efficient recovery of oil from bituminous sand.

DR. C. S. BEALS (Dominion Astronomer, Department of Mines and Technical Surveys): How do reserves of these tar sands compare with reserves of oil in western Canada?

DR. G. S. HUME (Director General of Scientific Services, Department of Mines and Technical Surveys): The bituminous content of the sands has been variously estimated at anywhere from 100 to 250 billion barrels. Presently established reserves of oil in Alberta are estimated at 1.3 to 1.5 billion barrels. With regard to the tar sands, it should be noted that, in a westerly or southwesterly direction, the thickness of overburden increases, and although there may be exposures in deep valleys and along the main river courses, the overburden in general is so thick that strip mining would be impossible: mining would have to be by costly underground operations. This is a factor that must be kept in mind when considering availability of reserves.

DR. BEALS: How does the cost of producing gasoline by refining the tar sands compare with that of producing it by hydrogenation of coal?

DR. WARREN: We have no reliable figures on the cost of producing gasoline from coal. My own impression is that, in the present state of development of such processes, it would be less expensive to produce oil or gasoline from the bituminous sands than from coal.

DR. D. S. MONTGOMERY (Fuels Division, Mines Branch, Dept. Mines and Tech. Surveys): From what Mr. Djingheuzian has told us, it seems evident that it is relatively easy to separate the bitumen from the sands by the cold water process, but that the product obtained is an extremely low grade crude oil which requires extensive refining to convert it into a marketable No. 2 fuel oil. Could Mr. Djingheuzian give us the approximate fraction of the cost that is involved in the separation process, assuming that the bitumen is to be marketed at Superior as No. 2 fuel oil at $3.50 a barrel?

MR. DJINGHEUZIAN: I can give only a very approximate estimate of the cost as this will depend very largely on the capacity of the plant to be used and therefore cannot be established until runs are made with a semi-commercial pilot mill. However, I would judge that the total operating cost per barrel of oil would be about 74.4 cents. This estimate includes all charges for taxes, insurance, etc., and depreciation spread over ten years, and is, I believe, high rather than low. It would constitute 21.3 per cent of the value of the final product.

DR. ELMER E. PEHRSON (U. S. Bureau of Mines): What is the recovery of oil in the cold water process?

MR. DJINGHEUZIAN: 97½ per cent of the total oil input, or 95 per cent of the bitumen when the diluent loss is expressed in terms of bitumen.

DR. PEHRSON: What is the quantitative recovery, that is, the amount of final product recovered per cubic yard?

MR. DJINGHEUZIAN: Just over one barrel. Mr. S. M. Blair, in his report, estimates a recovery of 20,680 barrels of oil per 20,000 cu. yd. of bituminous sands.

DR. PEHRSON: How far do you think operating costs of the cold water process can be reduced from the figure given by you?

MR. DJINGUEUZIAN: That I cannot say, because the costs will depend so much on the capacities of the machines used. In my estimate, I assumed ball mill capacities varying as the square of the diameter, but the factor might be 2.4 or even 2.6; for some mills operating in Denver it is said to be as high as 3.0. If capacities were of this order, both capital and operating costs

*January 1952 BULLETIN, pp. 2-15; Transactions, Vol. LV, 1952, pp. 1-14.

would be considerably lower than given in my estimate.

Dr. P. E. Gishler (National Research Council): How low can you bring the water content of the crude oil?

Mr. Djingueuzian: With the use of suitable wetting agents we can reduce it in a very short time to 21 or 22 per cent — provided the ratio of bitumen to diluent is 50:50 by weight. We ran one series of tests in which the ratio was 62.7:37.3. The thickener overflow assayed 50.5 per cent oil, 5.2 per cent mineral matter, and 44.3 per cent water. Thus the attempt to obtain a suitable crude oil for the heat treatment plant with such a high content of bitumen in the mill feed was a failure.

Dr. Gishler: In our process we are chiefly concerned with getting the water content of the crude oil as low as possible; the mineral matter can be very high. How do you go about eliminating the water?

Mr. Djingheuzian: In our tests, our purpose was to bring the water content to below 30 per cent and, even more important, the mineral matter content to below 5 per cent. Any mill runs in which this latter figure was exceeded or even closely approached were considered failures. Perhaps Dr. Montgomery could give us some information on the means taken to eliminate water. He carried out quite exhaustive laboratory investigations on that phase of the work.

Dr. Montgomery: It was found that both the *pH* of the water and the nature of the wetting agent had a profound influence upon the settling rate of water droplets in a mixture of equal parts by weight of Abasand bitumen and kerosene, due to the influence of these factors upon the coalescence of the drops of water. Briefly, the laboratory work strongly suggested that, by the selection of suitable wetting agents and by controlling the *pH* of the water and the temperature, tanks can be designed to reduce the water content of the oil phase to less than 5 per cent after 24 hours' settling.

DISCUSSION

ON

Review of Practices at the Sullivan Mine*

By J. R. GIEGERICH

(*Transactions, Volume LV,* 1952, pp. 336)

Mr. E. P. Thompson (Hollinger Consolidated Gold Mines, Ltd.): Would the author explain the paradox in the paper involving the two different sized holes, both of which are claimed to be the cheaper?"

Mr. Giegerich: Small diameter holes are drilled with carbide bits in development and small headings; the larger diameter diamond-drill holes were used for production. Each is the cheaper in its own field.

Mr. Thompson: We have found, at Hollinger, that safety meetings conducted by the Safety Engineer violated the principles of sound organization, and the result was that line supervisors tended to let the Accident Prevention Unit look after the function of accident prevention. As a result of this experience, we have changed our procedure and now the Mine Captain (line supervisor) is in complete charge of his meetings, and the Safety Engineer acts in a staff capacity only. I wonder if Mr. Giegerich would care to comment on this problem of organization. It is primarily one of line-staff relationship.

It might be of interest to mention that Hollinger is experimenting with classes in public speaking for its Supervisors for the purpose of stimulating communications. The results to date have been very gratifying.

Mr. Giegerich: We originally found that monthly safety meetings were overshadowed by safety measures, therefore we divided the meeting into two parts; (1) Safety, and (2) Production — with a time limit on each. Each Mine Captain had his own group and meetings; general meetings were held every month.

Mr. M. A. Twidale (Helen Mine) asked (1) if conveyors carried the 1½-inch material from the second-stage crushing; (2) how is dust controlled during second-stage crushing; and (3) is there any water problem during conveying.

In his reply, Mr. Giegerich said that (1) the ore conveyed is crushed to 3-inch size — it is crushed to second stage on the upper levels; (2) dust is controlled by direct ventilation outlets to surface; (3) as the ore is relatively dry, there is little or no water problem during conveying, but binder clay from fill may give some trouble.

Mr. C. S. Gibson (Mines Accident Prevention Association of Ontario, Timmins): What is the Union participation in the Suggestion Award Committee?

Mr. Giegerich: The Committee consists of twelve members, six appointed by the Union and six by the Management, one of the latter acting as Chairman. The Union group is changed every year.

Mr. A. A. McCloskey (Renabie Mines, Ltd.): What is the dimension of core and shell in pillar extraction?"

Mr. Giegerich: The shell for large pillars is 20 feet in thickness, the core is the remainder. No trouble is encountered in fragmentation of the shell.

*April 1952 Bulletin, pp. 198-203; *Transactions*, Volume LV, 1952, pp. 144-149.

Exploration, Development, and Production Practices at United Keno Hill Mines, Limited

*By H. BRODIE HICKS**

(Ottawa General Meeting, January, 1952)

(Transactions Volume LV, 1952, pp. 337-347)

ABSTRACT

The Mayo-Keno Mining District has grown from a small high-grade camp into one of the most important base-metal production centres in Canada. The operations of United Keno Hill Mines, Limited, have, to date, been by far the most important factor in this development.

The geographical location of the Keno Hill area is remote from any large centre of population and the climate is severe. Impetus to exploration was given by the Klondyke 'gold rush', since which time the area has been gradually prospected and developed.

Geologically, the country is an extension of British Columbia. The rocks in the immediate area are Precambrian. Structural control has not been fully worked out but progress is being made. Possibilities for discovery of new orebodies are excellent and extension to depth appears probable. The ore is exceptionally high-grade.

Mining in general follows conventional methods except that very heavy timbering is required due to the soft, friable nature of the vein material. Chief production to date has been won from the Hector mine, which has been developed to a nominal depth of 400 feet with a winze now being sunk to develop a further 500 feet.

The ore is concentrated by flotation, with production of a lead and a zinc concentrate, the latter carrying about one per cent cadmium. A cyanide plant, to recover silver values in the flotation plant tailings, has recently been added. Water supply is a serious problem.

Transportation facilities, very meagre in the past, are being improved by new road construction. United Keno Hill Mines operates its own truck fleet between Whitehorse and the concentrator. Handling of concentrates and handling and warehousing of incoming freight is a major factor in operations.

Power production is by Diesel engine, with a government-financed hydro-electric plant scheduled to be completed by the end of 1952.

The Company operates a number of services not normally part of a mining operation, due to the remote location. These include a complete logging and lumbering operation.

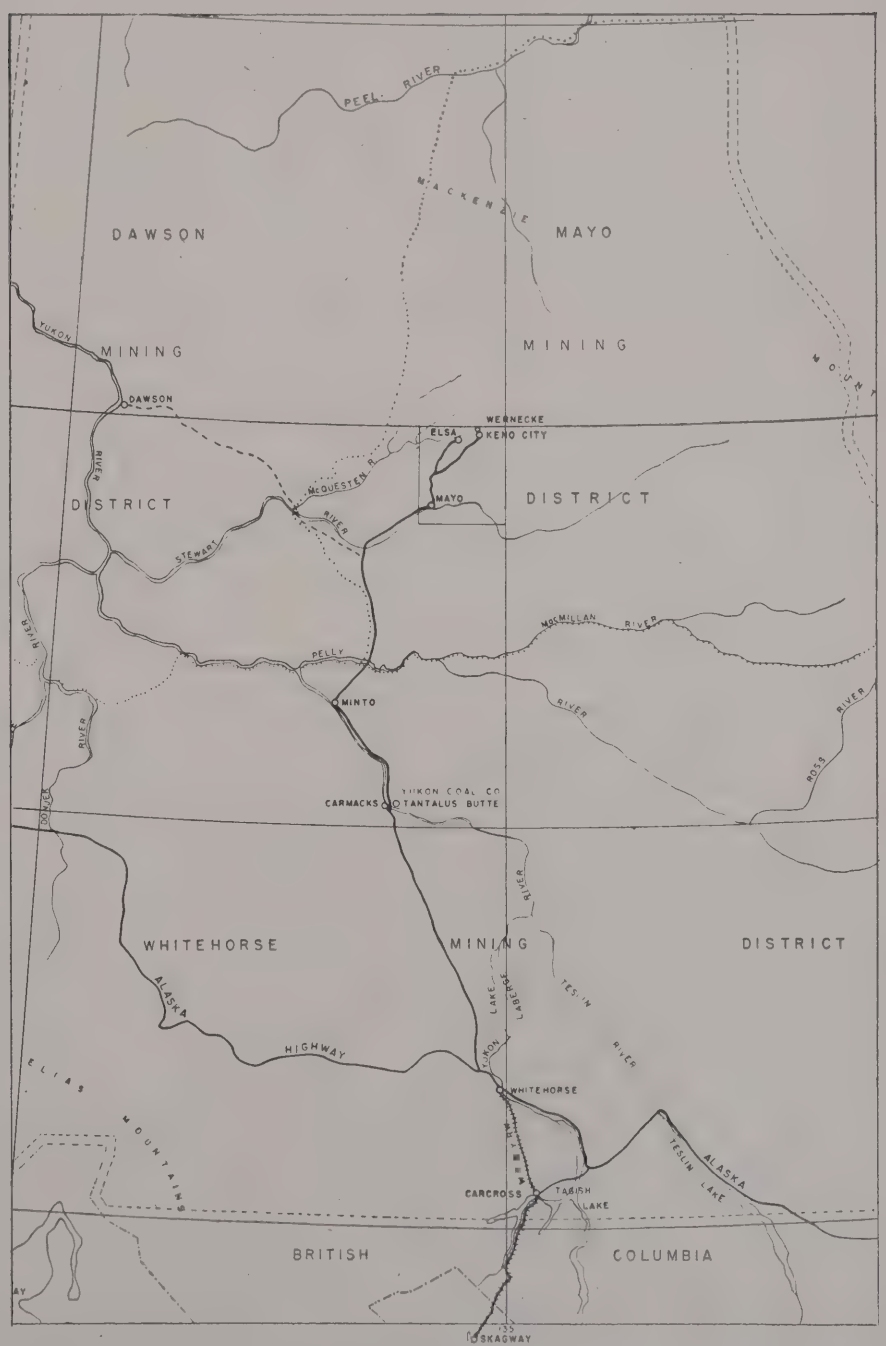

Figure 1.—Yukon Territory, showing location of properties of United Keno Hill Mines, Limited.
Scale: 1 in. = 95 miles

*General Manager, United Keno Hill Mines, Limited, Elsa, Y.T.

The Company also produces all coal mined in Yukon Territory through a subsidiary, the Yukon Coal Company. The deposits worked are on the Yukon river, near Carmacks.

* * *

Introduction

THE REVIVAL of the Mayo mining camp, with particular reference to the activities of United Keno Hill Mines, Limited, has been one of the more spectacular and also one of the most important of the post-war developments in the Canadian mining field. The geological location of the camp, coupled with other natural disadvantages, have in the past limited the mining area to small-scale, high-grading operations, but the post-war improvement in metal prices has now broadened the possibilities and has led to the considerable success achieved by United Keno Hill Mines, Limited, and the present boom-type development by a whole host of newcomers to the field.

Location

Yukon, which has an area of 207,000 square miles, is showing ever increasing possibilities as a mineral source of great value and more and more attention is being focussed on the Territory. The accompanying generalized map (Figure 1) shows the main transportation routes and mining districts.

The Mayo Mining District comprises roughly the eastern two-thirds of Yukon Territory lying north of McMillan river and south of the Peel. The district was one of the first in the Territory in which placer gold was discovered, but the occurrences proved of minor importance. Other minerals reported include ores of tungsten, tin, antimony, and several other metals, but of primary importance have been the ores containing silver-lead-zinc-cadmium now being developed and mined in the Keno area. Similar ores have been found over a very considerable area, extending some 50 miles north, 15 miles east, 30 miles west, and 10 miles south of the townsite of Keno Hills, Keno City, a total area of 2,700 square miles.

Present mining activity, centred on Keno City, is in an area on the southern flank of the MacKenzie mountains, an extension of the Rocky Mountain system which here runs in an almost east-and-west direction. Galena and Keno hills are 5,000 and 6,500 feet in height, respectively, and rise to elevations of some 2,000 to 3,000 feet above the floors of the surrounding valleys. Slopes are, in general, moderate, with a few notable exceptions, and road building and maintenance would be a matter of no great difficulty but for the presence of permafrost, with its various attendant problems.

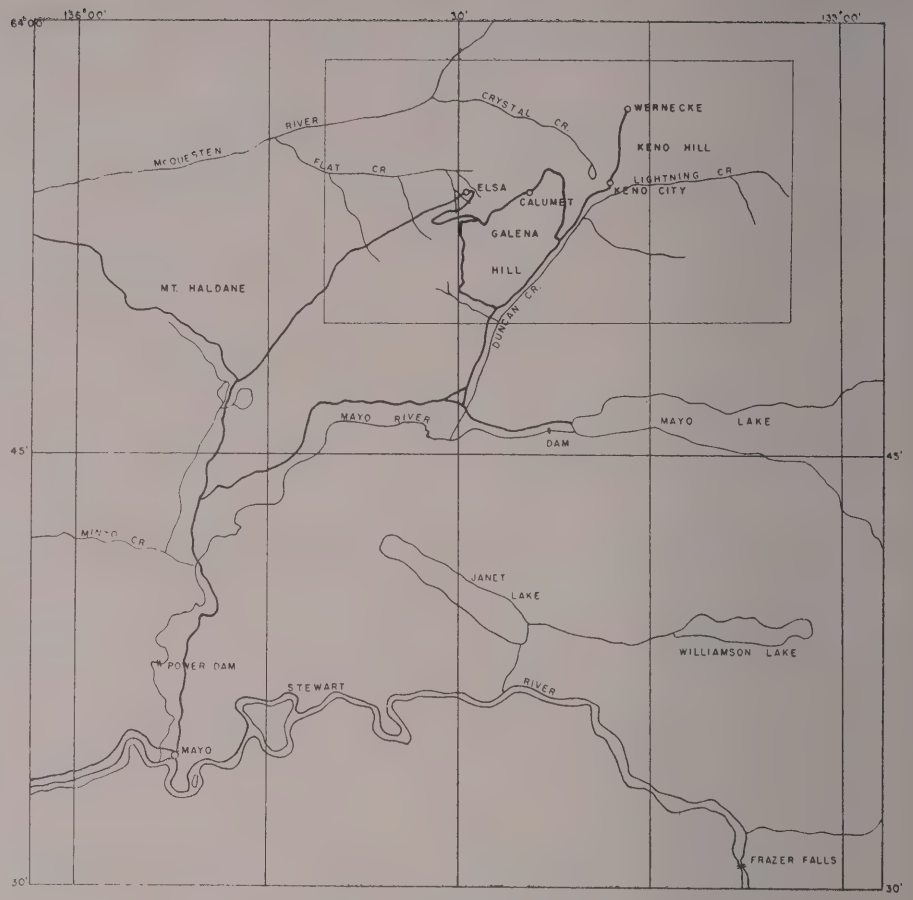

Figure 2.—Mayo-Keno area.

Scale: 1 in. = 8 miles

In a latitude some 150 miles below the Arctic Circle and at elevations between 3,000 and 6,000 feet, severe climatic conditions are encountered. Winters are long, dark, and cold. The mines are completely hidden from the sun for ten weeks each year, a fact which has a greater effect on morale than on any physical aspect of operation. Temperatures may be excessively low and a minimum of 81° below zero was officially recorded in Mayo during the winter of 1947. As higher elevations are attained, the winter temperature tends to moderate. Precipitation is light, amounting to an average of 11 inches per annum, and the area is generally considered as semi-arid.

Run-off of surface waters amounts to about 65 per cent, with severe fluctuations in river and water-table levels which have important consequences in navigation, in the development of water power, and in mining.

Figure 2 is a map of the general Mayo-Keno area with the present active mining area blocked out. Figure 3 is a claim map of this latter area, showing the present holdings of United Keno Hill Mines, Limited.

History

The first discovery of silver-lead ore was staked as the 'Silver King' claim in the canyon of Galena creek on the hill of the same name in 1906. This claim, incidentally, became a major producer in the 1930's.

Several other claims had been staked in the area to the north and east of the present producing centre. In 1919, Louis Beauvette discovered the first ore on top of Keno hill and, within a remarkably short time, nearly the whole hill had been staked.

The first commercial production, from a group of claims nearly on the top of the hill, was undertaken by Keno Hill, Limited, a subsidiary

Figure 3.—Map showing location of properties, Keno-Galena Hill area.
Scale: 1 in. = 15,000 ft.

of the Yukon Gold Company, Limited, one of the larger placer operators of the Dawson district. Costs were high, however, and, when the original high-grade ore shoots appeared exhausted, the project was abandoned.

In 1921, the Treadwell-Yukon Company, operated by the Bradley interests of San Francisco, acquired leases from Keno Hill, Limited, and pursued a vigorous policy in taking up ground on its own account. In the course of the next twenty-one years, Treadwell-Yukon accounted for better than 80 per cent of the total production of the district prior to the advent of the United Keno Hill Mines. In 1942, however, general world conditions, and in particular the lack of labour and low metal prices, led to the suspension of operations.

In 1946, the entire assets of the Treadwell-Yukon Company were acquired by United Keno Hill Mines, Limited, who have subsequently, by purchase and by staking, added greatly to the original holdings. This policy continues. The success achieved by this Company has led to a great resurgence of staking and prospecting in the area which has resulted in the extension of possible favourable ground considerably beyond anything formerly visualised. Interest is also reviving in nearly-forgotten prospects more remote from the present centre of activities.

Geology

Tht rocks of the area are classified as late Precambrian and consist chiefly of a thick series of interbedded schists and quartzites locally intruded by comformable greenstones. These greenstones, rather than representing sills of almost regional extent as previously supposed, appear upon further work to represent, in many cases, laccolithic intrusions along beds. A few acid dykes and sills have been mapped. Normally, the formation strikes a little north of east and dips thirty degrees to the south, but the structure is much broken by faulting and folding. The structure of the ore-bearing deposits has been the subject of much discussion and the problem is not yet solved, although recent work is doing much to bring a solution closer. Veins are usually referred to as of two types, the longitudinal and the transverse. The longitudinal veins are of regional dimensions, having been traced in some instances for nearly two miles with the probability of much greater extension. They strike northeasterly and normally dip about 70 degrees to the south. The transverse veins generally strike more northerly, with an irregular but usually steep dip. They are invariably related to one or more of the longitudinal veins.

It is the writer's opinion that ore is to be sought wherever local conditions have been favourable to the production of one or more transverse veins along the strike of or between two of the longitudinal veins. Such a condition prevailed on the Top-of-the-Hill claims, originally worked by Keno Hill, Limited, and a similar pattern is evolving as more work is done at the Hector mine and at the Calumet mine.

The country rock, while normally mapped distinctively as 'quartzite' or 'schist' is, in fact, invariably a bedded series of the two rock types with individual beds of either type seldom exceeding a few feet in thickness, and the rocks should be designated rather as 'predominantly quartzite' or 'predominantly schist', as the case may be.

Chief ore mineral is galena, which when unassociated with other minerals, carries from one to three hundred ounces of silver to the ton. Less common, but important, is freibergite (argentiferous tetrahedrite), picked specimens of which may assay as high as 8,000 ounces of silver to the ton. In some of the mines, oxidation is pronounced and a variety of secondary silver minerals have been identified. Zinc occurs fairly generally in the form of sphalerite and, at the Hector mine at least, the pure sphalerite carries about 1 per cent cadmium. Gold values average 0.01 ounces per ton, although considerably higher values have been encountered in several undeveloped prospects on Keno hill, notably at the No. 6 vein and the King vein.

At the present time, veins are being mined from 18 inches in width up to better than 30 feet, with an average in the neighbourhood of 8 or 9 feet. Greater widths than the

above have been reported from past operations.

The fact that, in the past, most of the local mines have been worked only to shallow depth has given rise to considerable discussion regarding the general depth possibilities of the camp. Consideration of individual cases, however, reveals that the reason for cessation of operation has almost invariably been one or another of several factors other than ore character, the most common being inability to cope with water once permafrost has been penetrated, or lack of power for hoisting from more than a moderate depth. In one or two instances where there does appear to have been a diminution of values, structural and depositional features remain unchanged and the probability that the condition represents merely the bottoming of an individual oreshoot is a strong one, which has never yet been put to the test of further lateral exploration. The ore is not of secondary origin and, as the outcrops cover a vertical span of some 3,000 feet, there is no reason to suppose that at least a succesion of veins may not extend to a considerable depth in any one mine.

At the Hector mine, more work of a purely exploratory nature has been performed than at all other mines of the district combined, and the bottom level is fully as productive as any other. This has led to the decision to sink a winze to open a further four levels at 125-foot intervals. At the half-way mark, the sinking was stopped to permit driving of cross-cuts to the No. 3 vein zone (a transverse vein), and on both levels typical ore, of normal width and grade, was encountered. The lower of these two levels is at a depth of roughly 700 feet below the outcrop. This is the greatest depth yet attained in the district and hence the ore occurrence here is of prime significance.

Figure 4 shows a plan of the 300 level of the Hector mine, with possible structure suggested. The same structure may ultimately be extended to the Calumet mine, which lies to the northeast on the opposite side of a heavy fault.

Exploration

Frost action on the mountain sides has at times the effect of bringing to the surface material from the bedrock below, even from considerable depths. When a piece of vein material — usually in the form of oxidised carbonate but sometimes, and preferably, more or less unaltered — has been found, the practice has been to sink a series of test pits, working uphill until no further float is found. The source is thus indicated and, by sinking further pits, the vein may be uncovered. Location is complicated by the fact that the bedrock, particularly on the steeper slopes, may show considerable creep, extending downward a number of feet and thus shearing off the top of the vein. The discoverer of the Hector deposit is reported to have spent eleven years digging pits between his first discovery of float and location of the No. 3 vein.

With the considerable number of prospectors roaming the hills in the 20's and early 30's it is probable that all deposits which could reasonably be located in this manner have at least been identified. It is worthy of note that all presently known mines have been discovered in areas of relatively shallow overburden — in creek beds or above timberline.

A number of the companies, realising the diffficulties of direct exploration from surface, have taken the course of driving adits into favourable ground, and in one or two instances they appear to be obtaining favourable results from this policy.

Geophysical prospecting is being experimented with to determine the most applicable method.

Diamond drilling from surface is being currently attempted by two companies. Considerable difficulty is being experienced in coring and in prevention of caving, but techniques are being worked out that should overcome these troubles. Underground drilling by United Keno Hill has revealed similar difficulties and reliance is usually placed largely on sludge samples. Sludge assays are not accepted quantitatively, but the presence of silver values is taken as an indication of a favourable zone. Much ore has been found in this way at the Hector and Elsa mines. United Keno Hill Mines is in a position rather different from that of other companies in the area as it has had a number of semi-developed properties to explore before having to turn its attention to its large holdings of virtually unexplored ground. These holdings include some twenty possible mines, in various stages of development, extended over a length of 14 miles and a width of 3½ miles. This represents a district rather than an individual mine, and this fact should not be lost sight of when past and future development is considered.

The Company had intended pursuing a vigorous policy of development on a number of these properties simultaneously but the rapid and unprecedented success achieved at the Hector mine has been such as to limit available resources for other work. A total of seven properties have, however, received at least superficial underground examination, all with encouraging results.

Figure 4 shows development at date on the 300 level of the Hector mine, where slightly more work has been performed than on either of the other two developed levels. Drifting along the No. 3 vein was about complete when a series of drill holes intercepted No. 4 vein. Had these holes not been drilled, it is uncertain whether the No. 4 would have been recognized at the juncture, due to the considerable distortion of the country rock. Veins No. 4 and No. 6 were at first considered to represent the same 'break' but later work on other levels gives the structural pattern represented. To the north, there appears to be another break, roughly parallel to the No. 6, indicated in drill holes and, somewhat distorted, in the drift near the major cross-fault zone. It is not a particularly wide, or strong, break and shows only low values where drilled to date. In line with the possible structural basis discussed above, this break and the No. 6 would be identified as belonging to the 'longitudinal' type of vein structure, and No. 3 and No. 4 veins to the 'transverse' type. The Jock vein is known from old surface pits and recent drilling and shows typical structure with some values. It presumably represents a 'transverse' vein on the other side of the No. 6 structure.

On this hypothesis, future work will consist of exploration of the No. 6 break, with drilling into both walls with a view to picking up transverse veins as they occur.

It is of interest to note that, of the drifting to date on this level, 100 per cent of that on No. 3 vein, 83 per cent of that on No. 4 vein, and 40 per cent of that on No. 6 vein, has been in ore assaying over 25 oz. silver per ton. None of the drifting has fallen below the 5 oz. mark.

A good proportion of the properties in the area can be opened to considerable depths by means of adits, but there are some areas on both Keno and Galena hills where the slope is insufficiently steep to permit development by means of adit and where shaft sinking is

Figure 4.—Hector mine, 300 level, showing longitudinal (L) and transverse (T) veins.
Scale: 1 in. = 585 ft.

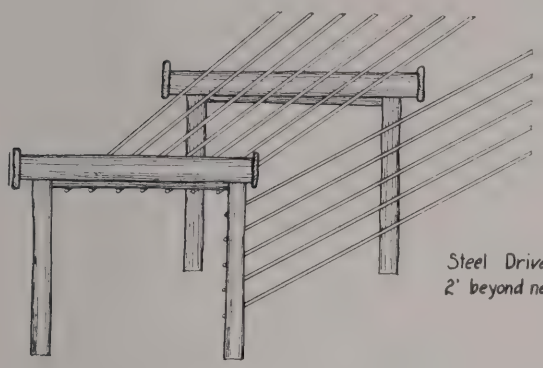

Figure 5.—Use of spiling steel.

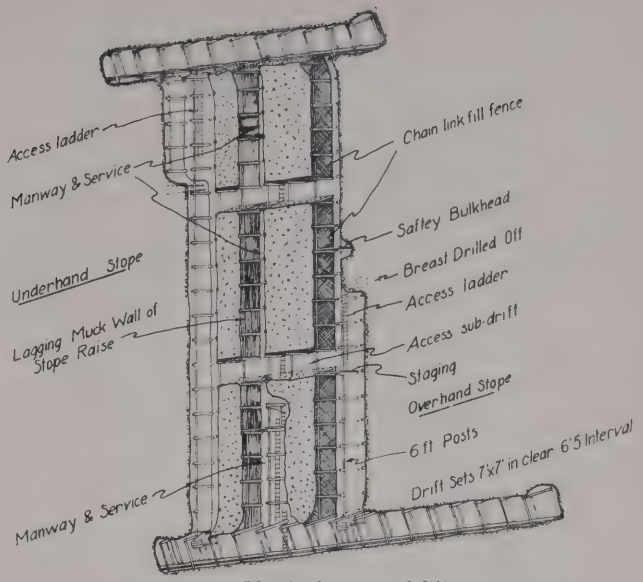

Figure 6.—Vertical cut-and-fill stope.

planned. No sinking from surface has as yet been attempted but one such shaft is planned at the No Cash mine.

In the following descriptions of development methods, it should be borne in mind that rock conditions vary greatly from place to place throughout the district: in hardness, from very hard to mud; and in strength, from self-supporting to ground requiring spiling on all sides. In general, the softer, weaker types predominate. This difference leads to marked variation in mining methods and in equipment employed. In a paper of this scope, only one or two examples can be given.

At the Onek mine, for instance, the main development adit, which was driven this past summer for a distance of 1,314 feet through ground of medium hardness and strength and nearly flat bedding, made an advance averaging nearly 300 feet per month. The heading has been driven 7 ft. by 7 ft. inside the timber, with local spruce sets at 10-foot intervals. Posts are 10-inch tops and caps 12-inch tops, and sets are invariably lagged on the back and frequently on the hanging-wall also. Lagging is open to permit relief of subsequent pressure should it develop. Direct costs of the adit are given in Table I.

In drifting, by far the greatest experience has been gained at the Hector mine, where the vein material is, in general, very soft and weak. Headings are driven 7 ft. by 7 ft. and later slashed to full vein width if necessary. The method of driving is similar to that described for the adit but much more timber is required; and, while set spacing of 6½ feet is standard, this is frequently decreased in practice. Backs and hanging-wall are invariably lagged and blocked tight, as also is the footwall occasionally. The cycle is to muck out, timber tight to the face, then drill and blast.

TABLE I.— DIRECT COSTS, MAIN ADIT DRIVE, ONEK MINE
Footage — 1,314 feet

Supervision and engineering	$ 1.34
Operating labour	14.85
Operating supplies	0.54
Repair supplies	0.48
Drill maintenance and bits	2.47
Pipe lines and hose	1.11
Explosives	3.57
Timbering materials	2.16
Haulage, underground and surface	3.90
Pumping and ventilation	0.27
Compressor operation	5.71
	$36.40

Overbreak in the soft ground has always presented a problem, but conventional spiling methods, except under the worst circumstances, have usually proved a remedy more costly than the disease. A newly developed method, which appears very successful, is to drill-in a number of pieces of drill steel as spiling. Under ordinary circumstances, these are usually 12 feet long and spaced about one foot apart. The method is shown in Figure 5. The cost in steel is high but is outweighed by the saving in labour; moreover, much of the steel should be recoverable ultimately. At the present time, a cheaper substitute for conventional drill steel is being sought. The method has been successfully applied not only to drift backs but to backs and walls in all types of opening.

TABLE II.— DRIFTING COSTS, HECTOR MINE, 1951

Power	$4.22	per ft.
Direct mining supplies	3.18	"
Timber	2.58	"
Drilling labour	2.41	"
Mucking labour	3.59	"
Timbering labour	2.24	"
Bonus	4.94	"
General supplies	1.67	"
Loader and Mancha trammer (excluding labour)	2.57	"
DIRECT COSTS	$27.40	"
ESTIMATED DISTRIBUTABLE COSTS	$ 5.50	"
TOTAL COSTS	$32.90	per ft

Table II shows current drifting costs, and Table III labour distribution in various types of mine workings, at the Hector mine in 1951. Table III has been included to emphasize the high proportion of time necessarily devoted to timbering.

Raises are normally placed at 100-foot intervals along the vein and are driven straight up the dip. In better ground, a stulled type of timbering is used for widths up to 12 to 14 feet. For greater widths, cribbing is employed. In bad ground, cribbing is used without regard to width. In stulled raises, the hanging-wall is invariably lagged and blocked tight; frequently, this is done also

TABLE III.—LABOUR DISTRIBUTION, HECTOR MINE, 1951

	DRIFTING	RAISING	STOPING	ALL WORK
Drilling	29.3%	26.8%	19.1%	21.8%
Mucking	43.6%	20.1%	20.9%	24.4%
Timbering	27.1%	53.1%	53.6%	49.2%
Filling	—	—	6.4%	4.6%
	100.0%	100.0%	100.0%	100.0%

on both side walls. Two men with one stoper form a standard crew and these men also install the timber. The raise is usually started on the permanent drift timber and a standard muck chute is employed. Raising costs are given in Table IV and stoping costs in Table V.

Shaft sinking is under way at the Hector mine. The shaft is in solid footwall quartzites where no special problems have so far been encountered. It is a vertical, three-compartment opening, with two compartments 5 ft. by 6 ft. and one 4 ft. by 6 ft.

In all future shaft sinking, the question of location with regard to a suitable rock type will receive primary consideration.

All production of importance to date has been won from the Hector mine, which is opened by a 1,600-foot adit driven at a nominal elevation of 400 feet below the outcrop. The Calumet mine is developed through the same adit but lies 400 feet farther in. Two other levels have been opened in the Hector mine at approximately equal intervals above the adit level.

Supplies to the upper two levels are handled through a small service raise, equipped with timber slide.

Both past operators and the present Company have experimented with various types of stoping, with the result that the present Company is employing two methods. In orebodies over 14 feet in width, the stoping system resembles the normal intermittent cut-and-fill method except that no attempt is made to salvage timber, and each new set is placed on the cap of the preceding set before mucking is undertaken. In narrower orebodies, a system of vertical slicing, illustrated in Figure 6, has been introduced.

In this system, a two-compartment raise is first driven through the centre of the stope, following which vertical slices are carried up each side of the raise from level to level. The width of the slice depends on the character of the ground being

TABLE IV.—RAISING COSTS, HECTOR MINE, 1951

Power	$2.82	per ft.
Supplies	1.93	"
Timber	5.02	"
Drilling labour	2.22	"
Mucking labour	1.66	"
Timbering labour	4.40	"
Bonus	5.73	"
DIRECT COSTS	$23.78	"
ESTIMATED DISTRIBUTABLE COSTS	$ 5.65	"
TOTAL COSTS	$29.43	per ft

TABLE V.—STOPING COSTS, HECTOR MINE, 1951

Power	$0.36	per ton
Supplies	0.19	"
Timber	0.92	"
Drilling labour	0.42	"
Mucking labour	0.46	"
Timbering labour	1.18	"
Filling labour	0.14	"
Bonus	0.94	"
DIRECT COSTS	$4.61	"
ESTIMATED DISTRIBUTABLE COSTS	$1.50	"
TOTAL COSTS	$6.11	per ton

mined. Close timbering is carried at all times. An attempt is made to provide continuous production through mining opposite sides alternately. After a slice has been carried to completion, it is lagged off and back-filled. Access to the next slice is maintained by means of carrying timbered 'drifts' through the fill from the original raise.

A novel feature of the fill wall is the use of wire-mesh fencing in place of timber lagging; this has reduced fill-fence construction from 28 man-shifts to 8 man-shifts.

Recently, experiments have been started in underhanding the slices and, at the moment of writing, the new method appears to be resulting in decreased overbreak. Horizontal holes are used as illustrated in Figure 6, due to the difficulty of drilling vertical holes and keeping them open.

Fill for the upper-level stopes is obtained in summer from the surface, by bulldozing gravel into the stope raises; during the colder weather it is obtained by driving openings into the stope walls. Where a length of continuous ore is being mined, the waste is supplied from raises driven into the footwall, but where the stope has ended against a body of waste, the fill is obtained by driving drifts along the vein into the waste section. This serves also as an exploratory heading and blind shoots of ore have thus been discovered. On the lower two levels fill is normally obtained from development headings.

Muck on the two upper levels is collected in trains of 20 cu. ft. Hammant rocker-dump cars, handled by Mancha 'Little Trammer' 1½-ton locomotives. It is dumped into either of two ore passes and pulled on the adit level into 30 cu. ft. rocker-dump cars usually handled by a Mancha 'Mule' 2½-ton locomotive or at times by one or more of the smaller machines. The ore proceeds in trains of up to twenty cars to the head-bin of the aerial tramway.

Air, at 100 lb., is supplied by means of an Ingersoll-Rand 850 c.f.m. compressor and a Belliss-Morcom 1,100 c.f.m. compressor; a third unit of 1,500 c.f.m. is on order. Water for drilling is obtained from mine seepage, which is collected in a sump on the adit level and re-pumped through the workings. In order to prevent stagnation and freezing during the winter, the high point of the system is fitted with a Morrison relief valve set at 60 pounds, which permits flow when all other outlets are closed. During the coldest weather, individual heading lines are drained between shifts.

Drainage water fluctuates widely with the season, being very heavy through the late spring and summer and drying up largely in winter, although a flow of some 30 g.p.m. remains at the lowest point. Due to low rock temperatures, the water is always near the freezing point and during cold weather this formerly caused a great deal of trouble through freezing back at the portal mouth. This has been overcome by installing a sump and pump about 500 feet within the adit mouth and beyond the influence of outside temperature. The water is trapped here and pumped through a small pipe at high velocity to beyond the entry. Of equal importance, however, is the necessity of supplying some heat at the portal house; this is restricted by ventilation doors from entering the mine.

At the Onek mine, where water for drilling is supplied from outside,

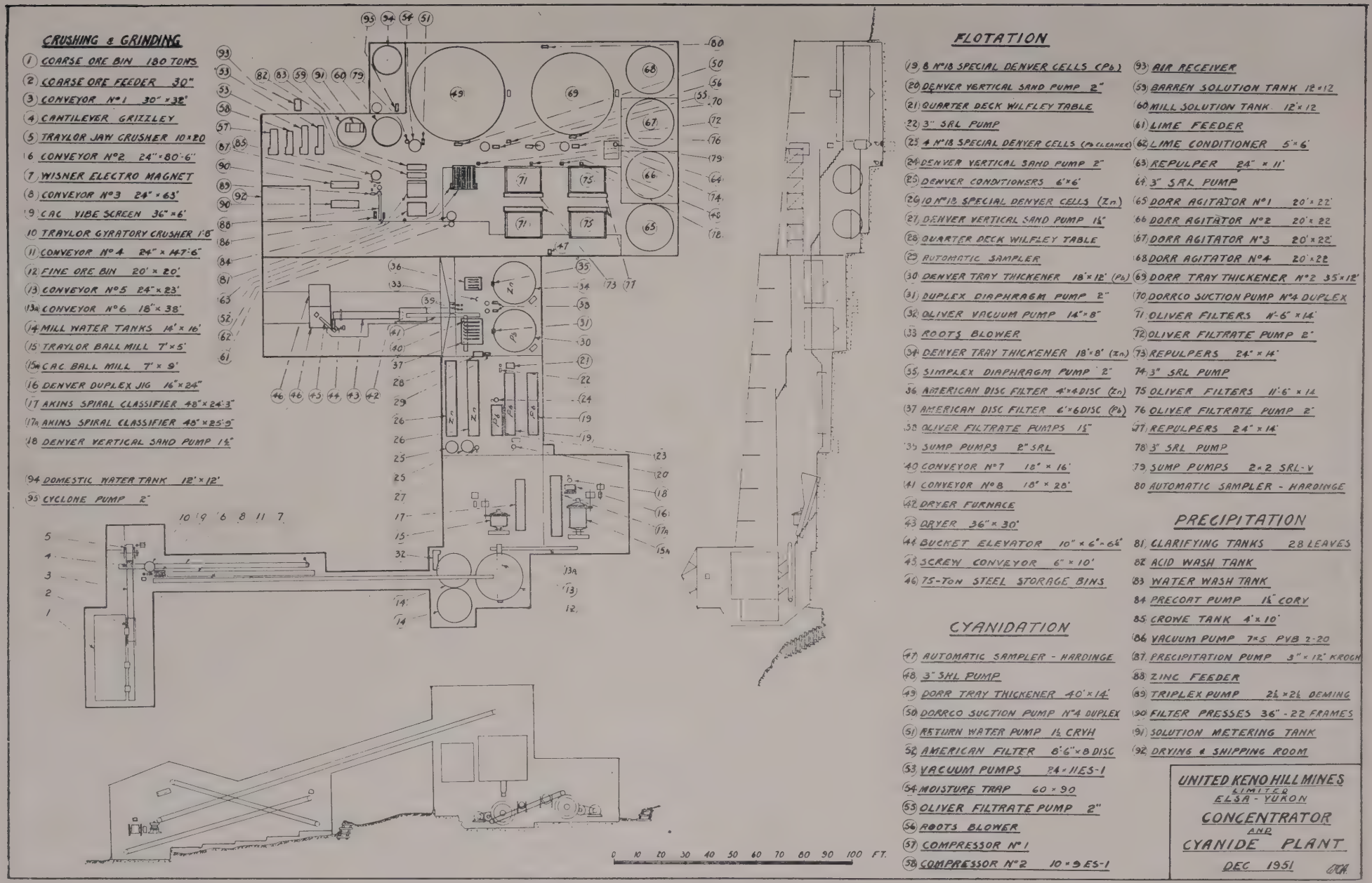
CRUSHING & GRINDING
1 COARSE ORE BIN 180 TONS
2 COARSE ORE FEEDER 30"
3 CONVEYOR N°1 30" × 32'
4 CANTILEVER GRIZZLEY
5 TRAYLOR JAW CRUSHER 10×20
6 CONVEYOR N°2 24" × 80'-6"
7 WISNER ELECTRO MAGNET
8 CONVEYOR N°3 24" × 63'
9 CAC VIBE SCREEN 36" × 6'
10 TRAYLOR GYRATORY CRUSHER 1'8"
11 CONVEYOR N°4 24" × 147'-6"
12 FINE ORE BIN 20' × 20'
13 CONVEYOR N°5 24" × 23'
13A CONVEYOR N°6 18" × 38'
14 MILL WATER TANKS 14' × 16'
15 TRAYLOR BALL MILL 7' × 5'
15A CAC BALL MILL 7' × 9'
16 DENVER DUPLEX JIG 16" × 24"
17 AKINS SPIRAL CLASSIFIER 48" × 24'-3"
17A AKINS SPIRAL CLASSIFIER 48" × 25'-9"
18 DENVER VERTICAL SAND PUMP 1½"
94 DOMESTIC WATER TANK 12' × 12'
95 CYCLONE PUMP 2"
FLOTATION
19 8 N°18 SPECIAL DENVER CELLS (Pb)
20 DENVER VERTICAL SAND PUMP 2"
21 QUARTER DECK WILFLEY TABLE
22 3" SRL PUMP
25 4 N°18 SPECIAL DENVER CELLS (Pb CLEANER)
24 DENVER VERTICAL SAND PUMP 2"
25 DENVER CONDITIONSRS 6'×6'
26 10 N°18 SPECIAL DENVER CELLS (Zn)
27 DENVER VERTICAL SAND PUMP 1½"
28 QUARTER DECK WILFLEY TABLE
29 AUTOMATIC SAMPLER
30 DENVER TRAY THICKENER 18'×12' (Pb)
31 DUPLEX DIAPHRAGM PUMP 2"
32 OLIVER VACUUM PUMP 14"×8"
33 ROOTS BLOWER
34 DENVER TRAY THICKENER 18'×8' (Zn)
35 SIMPLEX DIAPHRAGM PUMP 2"
36 AMERICAN DISC FILTER 4'×4 DISC (Zn)
37 AMERICAN DISC FILTER 6'×6 DISC (Pb)
38 OLIVER FILTRATE PUMPS 1½"
39 SUMP PUMPS 2" SRL
40 CONVEYOR N°7 18" × 16'
41 CONVEYOR N°8 18" × 28'
42 DRYER FURNACE
43 DRYER 36" × 30'
44 BUCKET ELEVATOR 10" × 6"-68'
45 SCREW CONVEYOR 6" × 10'
46 75-TON STEEL STORAGE BINS
CYANIDATION
47 AUTOMATIC SAMPLER - HARDINGE
48 3" SHL PUMP
49 DORR TRAY THICKENER 40' × 14'
50 DORRCO SUCTION PUMP N°4 DUPLEX
51 RETURN WATER PUMP 1½ CRVH
52 AMERICAN FILTER 8'-6" × 8 DISC
53 VACUUM PUMPS 24 × 11 ES-1
54 MOISTURE TRAP 60 × 90
55 OLIVER FILTRATE PUMP 2"
56 ROOTS BLOWER
57 COMPRESSOR N°1
58 COMPRESSOR N°2 10 × 9 ES-1
93 AIR RECEIVER
59 BARREN SOLUTION TANK 12 × 12
60 MILL SOLUTION TANK 12' × 12
61 LIME FEEDER
62 LIME CONDITIONER 5' × 6'
63 REPULPER 24" × 11'
64 3" SRL PUMP
65 DORR AGITATOR N°1 20' × 22'
66 DORR AGITATOR N°2 20' × 22
67 DORR AGITATOR N°3 20' × 22'
68 DORR AGITATOR N°4 20' × 22
69 DORR TRAY THICKENER N°2 35' × 12'
70 DORRCO SUCTION PUMP N°4 DUPLEX
71 OLIVER FILTERS N°-6' × 14'
72 OLIVER FILTRATE PUMP 2"
73 REPULPERS 24" × 14'
74 3" SRL PUMP
75 OLIVER FILTERS 11'-6" × 14
76 OLIVER FILTRATE PUMP 2"
77 REPULPERS 24" × 14'
78 3" SRL PUMP
79 SUMP PUMPS 2 × 2 SRL-V
80 AUTOMATIC SAMPLER - HARDINGE
PRECIPITATION
81 CLARIFYING TANKS 28 LEAVES
82 ACID WASH TANK
83 WATER WASH TANK
84 PRECOAT PUMP 1½" CORY
85 CROWE TANK 4' × 10'
86 VACUUM PUMP 7 × 5 PVB 2-20
87 PRECIPITATION PUMP 3" × 12" KROGH
88 ZINC FEEDER
89 TRIPLEX PUMP 2½ × 2½ DEMING
90 FILTER PRESSES 36" - 22 FRAMES
91 SOLUTION METERING TANK
92 DRYING & SHIPPING ROOM
UNITED KENO HILL MINES LIMITED
ELSA - YUKON
CONCENTRATOR AND CYANIDE PLANT
DEC 1951
0 10 20 30 40 50 60 70 80 90 100 FT.

Figure 7.—Concentrator and cyanide plant.

a novel method of prevention of freezing at the portal has proved successful. The water is circulated from the compressor room to a point in the adit beyond freezing temperatures and is there boxed parallel to the air line, the heat from which raises the temperature of the water markedly. Circulation is maintained when no water is being drawn by means of a relief valve discharging into the compressor cooling-water tank.

Underground temperatures are, in general, slightly below the freezing point and ice crystals may be found in freshly broken rock faces. However, this cannot be said to affect mining operations seriously. Apart from special precautions necessary with drilling water and drainage mentioned above, and the impossibility of storing any quantity of broken ore or waste-fill underground, disadvantages of the low temperatures are confined chiefly to such relatively minor matters as icy track, ice-covered timbers, and so forth, all of which may be overcome by care and planning.

Ventilation is natural, being upcast in summer and downcast in winter; but for a period in both spring and fall, circulation is poor. Dead-end headings are furnished with auxiliary ventilation by means of 16-inch Meco air-driven fans, type *CF4*, connected to 10-inch Minair collapsible tubing.

The aerial tram between the Hector and Elsa camps is nearly three miles in length and descends a distance of 1,300 feet. An angle has been introduced about one-third of the distance from the Hector to permit loading of the ore from the No Cash mine, and the remaining two-thirds is divided into two portions by a tension station. Bins at each end are heated and the full buckets do not become seriously frozen even in the coldest weather.

Milling

The original concentrating plant, inherited from the Treadwell-Yukon Corporation, was a bulk flotation unit. Concentrates were not credited for the contained zinc and cadmium. This plant was destroyed by fire in June, 1949, and was replaced in October of that year by the present differential flotation unit. Subsequently, extensive laboratory test work has shown the possibility of economic recovery of silver from the tailings by cyanidation and, accordingly, a cyanide unit of 350 tons daily capacity is being erected

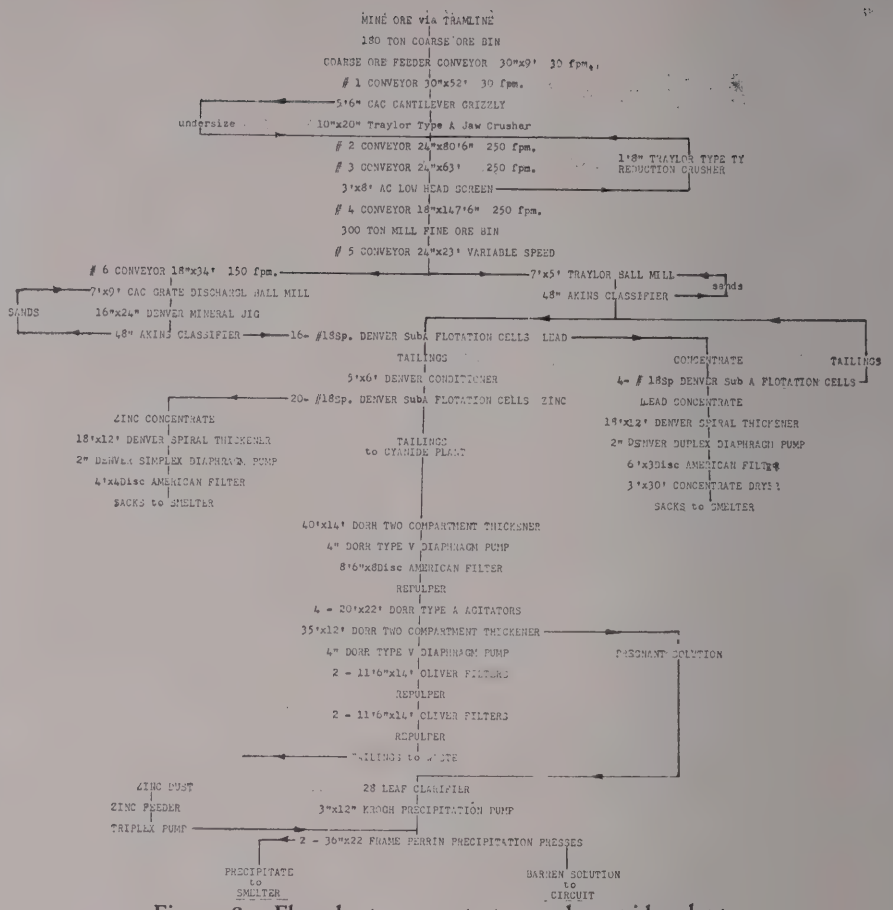
MINE ORE via TRAMLINE
180 TON COARSE ORE BIN
COARSE ORE FEEDER CONVEYOR 30"x9' 30 fpm.
1 CONVEYOR 30"x52' 30 fpm.
5'6" CAC CANTILEVER GRIZZLY
undersize
10"x20" Traylor Type A Jaw Crusher
2 CONVEYOR 24"x80'6" 250 fpm.
3 CONVEYOR 24"x63' 250 fpm.
1'8" TYLOR TYPE TY REDUCTION CRUSHER
3'x8' AC LOW HEAD SCREEN
4 CONVEYOR 18"x147'6" 250 fpm.
300 TON MILL FINE ORE BIN
5 CONVEYOR 24"x23' VARIABLE SPEED
6 CONVEYOR 18"x34' 150 fpm.
7'x5' TRAYLOR BALL MILL
48" AKINS CLASSIFIER
7'x9' CAC GRATE DISCHARGE BALL MILL
SANDS
16"x24" DENVER MINERAL JIG
48" AKINS CLASSIFIER
16- #18Sp. DENVER Sub A FLOTATION CELLS LEAD
TAILINGS
5'x6' DENVER CONDITIONER
CONCENTRATE
TAILINGS
4- # 18Sp DENVER Sub A FLOTATION CELLS
20- #18Sp. DENVER Sub A FLOTATION CELLS ZINC
LEAD CONCENTRATE
14'x12' DENVER SPIRAL THICKENER
2" DENVER DUPLEX DIAPHRAGM PUMP
6'x30 Disc AMERICAN FILTER
3'x30' CONCENTRATE DRYER
SACKS to SMELTER
ZINC CONCENTRATE
18'x12' DENVER SPIRAL THICKENER
2" DENVER SIMPLEX DIAPHRAGM PUMP
4'x4Disc AMERICAN FILTER
SACKS to SMELTER
TAILINGS to CYANIDE PLANT
40'x14' DORR TWO COMPARTMENT THICKENER
4" DORR TYPE V DIAPHRAGM PUMP
8'6"x8Disc AMERICAN FILTER
REPULPER
4 - 20'x22' DORR TYPE A AGITATORS
35'x12' DORR TWO COMPARTMENT THICKENER
DESCHANT SOLUTION
4" DORR TYPE V DIAPHRAGM PUMP
2 - 11'6"x14' OLIVER FILTERS
REPULPER
2 - 11'6"x14' OLIVER FILTERS
REPULPER
TAILINGS to DUMP
ZINC DUST
ZINC FEEDER
TRIPLEX PUMP
28 LEAF CLARIFIER
3"x12" KROGH PRECIPITATION PUMP
2 - 36"x22 FRAME PERRIN PRECIPITATION PRESSES
PRECIPITATE to SMELTER
BARREN SOLUTION to CIRCUIT

Figure 8.—Flow-sheet, concentrator and cyanide plant.

and is expected to be in operation early in December (1952).

Figure 7 shows the dimensions and general construction of the two plants, which are contiguous. The buildings are, in general, divided into 14-foot bays with trusses on 12 in. by 12 in. B.C. fir posts. All other lumber is local white spruce. Studding between posts is 2 in. by 6 in. sheathed outside with shiplap, one layer of tar-saturated felt, and one layer of Johns-Manville 'Asbestoside'. The roof is insulated throughout and the walls at a few points with 3-inch rockwall batts, sheathed inside with shiplap. Footwalls, retaining walls, and floor are concrete. Except in offices, the plants have no windows.

The flow-sheets of the flotation plant and of the cyanidation plant are shown in Figure 8 and are largely self-explanatory.

The coarse-ore bin, of 180 tons live capacity, is so constructed as to be able to receive ore from the aerial tram, directly from the Elsa mine by underground cars, or by truck delivery from other properties. The bin has three discharge chutes and is fitted with a movable feeder discharging onto a 30-inch belt. This belt delivers the ore over a 3 ft. by 5 ft. 6 in. Canadian Allis-Chalmers cantilever grizzly into a 10 in. by 20 in. Traylor jaw crusher. The product from this is elevated by two 24-inch conveyor belts to a 3 ft. by 6 ft. Canadian Allis-Chalmers vibrating screen. Screen oversize drops to a 1 ft. 8 in. Traylor gyratory crusher, discharge from which joins the jaw crusher product. Screen undersize is finished material and is conveyed to the fine-ore bin in the mill by means of an 18-inch belt, 148 feet in length.

The fine-ore bin is a wood-stave tank, 20 ft. by 20 ft., with a live capacity of 225 tons. Three outlets discharge onto a 24-inch conveyor belt, discharging directly into the ball-mill.

The ball-mill is a 5 ft. by 7 ft. Traylor, operating in closed circuit with a 48-inch Akins submerged spiral classifier.

A second ball-mill has now been installed in parallel with the original unit. This is a 7 ft. by 9 ft. Allis-Chalmers grate discharge mill, operating in closed circuit with a 48-inch Akins classifier and with a 16 in. by 24 in. Denver Duplex jig. Provision has been made for the installation of a second fine-ore bin to feed this mill directly, but in the meanwhile it will be fed from a cross-belt receiving the discharge of the present mill-feed belt.

Ball-mill discharge is split be-

tween two banks each of eight Denver No. 18 Special flotation cells, product of which is cleaned and recleaned in a further bank of four cells. The tailings are conditioned in a 6 ft. by 6 ft. Denver conditioner and then floated in a bank of ten Denver No. 18 Special cells for zinc recovery. Tailings from these go to the cyanide plant. A further bank of ten Denver No. 18 cells is erected and available for immediate expansion.

The lead concentrate is thickened in an 18 ft. by 12 ft. Denver two-compartment thickener, feeding a 6-foot 6-leaf American filter. The product is dried in a rotary-type dryer, sacked, and weighed. The zinc concentrate is similarly treated in an 18 ft. by 18 ft. Denver thickener and a 4-foot 4-leaf American filter but is not dried.

Tables VI and VII show operating results and costs of the flotation plant for the first six months of 1951. Reagents employed are set out in Table VIII. The cyanide plant follows conventional lines.

TABLE VII.— FLOTATION MILL OPERATING COSTS
First six months, 1951

Crushing and conveying	$0.28 per ton
Grinding and classifying	0.37 "
Flotation, thickening, and filtering	0.74 "
Sacking concentrates	0.99 "
General internal mill expense	0.36 "
DIRECT MILLING COST	$2.74 per ton

Reagent consumption is expected to approximate 3.5 pounds of cyanide per ton and 12 pounds of lime. With an expected cyanide head of 10 ounces, tails will be of the order of 2 ounces, without regrinding. Laboratory tests to date have indicated no benefit from regrinding but a plant-scale test, using the old 7 ft. by 5 ft. ball-mill is planned.

The Company's full-time metallurgist is devoting much test work to attempting to improve lead recoveries. The inability to float a portion of the lead and with it some of the silver appears to be due to the presence of various oxidation products of galena which do not respond to standard reagents. At present, attention is being given to experiments with the Buckman tilting table and these appear to hold some definite promise.

TRANSPORTATION

The programme of road building sponsored by the Federal government and to a lesser extent by the Territorial Government is gradually alleviating the difficulties in transportation which have hampered mining operations in the area. Present successful mining efforts not only justify past expenditures on this work but also warrant consideration being given to a greatly enlarged future programme.

The newly completed road from Whitehorse to Mayo Landing is a second-grade gravel highway, 258 miles long, to which must be added the further 32 miles from Mayo Landing to the United Keno concentrator at Elsa. The route of the roads may be traced in Figures 1 and 2.

The Whitehorse-Mayo road crosses three rivers, the Yukon, Pelly, and Stewart. These are served by ferry in summer and by ice bridges in winter, the latter consisting of a cunningly interwoven mat of logs laid on the earliest fall ice and subsequently built up by pumping water onto the surface. Unfortunately, due to the geographical history of the three rivers, their behaviour differs greatly, and there may be a considerable lag between freezing and thawing of one or another. This leads to interruptions of transportation of from six weeks to two months in spring and fall.

TABLE VI.— FLOTATION MILL OPERATING DATA
First six months, 1951

Dry tons milled	42,976.4 tons
Dry tons lead concentrate produced	4,451,5 "
Dry tons zinc concentrate produced	2,809.2 "

	SILVER	LEAD	ZINC
Mill heads	52.12 oz./t.	10.49%	4.99%
Tailings	13.06 "	3.27%	0.83%
Lead concentrate	338.20 "	70.80%	7.79%
Zinc concentrate	95.48 "	6.74%	53.49%
Recovery	79.18%	74.10%	86.19%

Oxide lead in final tailings	2.78%
Iron in zinc concentrate	5.20%
Cadmium in zinc concentrate	0.84%
Ratio of concentration	5.92

TABLE VIII.— FLOTATION REAGENTS

WHERE ADDED	REAGENT	POUNDS/TON
Ball mill	Lime	1.00
	Zinc sulphate	0.80
	Sodium cyanide	0.25
	Aerofloat 31	0.08
	Sodium sulphite	0.10
Classifier overflow	Aerofloat 31	0.08
	Reagent 343	0.07
	Cresylic acid	0.05
Zinc conditioner	Copper sulphate	0.45
Zinc cells	Pine oil	0.03
	Reagent 3	0.08

Maintenance of the road is shared between the Federal government and users of the road, the latter contributing through a system of tolls, a matter of considerable controversy. These tolls are set forth in Table IX and it should be noted that vehicle weight itself carries a toll which, in the case of concentrates, raises the total to approximately $5.25 per ton.

TABLE IX.— SCHEDULE OF TOLLS, WHITEHORSE-MAYO ROAD

Food for human consump'n	$2.00 per ton
General freight	3.00 "
Vehicle weight	1.00 "
Ores or concentrates	4.00 "

Incoming freight normally originates in Vancouver and proceeds by ocean steamer to Skagway, Alaska, thence by narrow-gauge railway to Whitehorse, and finally by truck to the mining property Outgoing concentrates follow the same route in reverse plus a further rail haul from Vancouver to Trail. Typical freight costs from Vancouver to mine are shown in Table X. Cost of concentrate shipment, Elsa to Trail, is approximately $50.00 per ton.

TABLE X.— FREIGHT COSTS PER 100 POUNDS, VANCOUVER TO MINE

Mining machinery	$3.95
Fruits and vegetables (perishables)	7.25
Potatoes, onions, turnips	6.00
Insulating materials	6.82
Ore sacks	3.30
Dynamite and high explosives	7.30
Tires and tubes	4.80

Transport Division

United Keno Hill Mines handles all transportation of its incoming freight and outgoing concentrates between the mines and Whitehorse on its own trucks. South-bound freight consists of a minimum of 20,000 tons of concentrate annually while north-bound tonnage normally amounts to between one-quarter and one-third of the south-bound tonnage, and includes fuel oil and coal from the Yukon Coal Company, at Carmacks.

Trucks employed at present are single rear-axle models, with a 269 cu. in. piston displacement engine, developing 88 horsepower at normal speeds, equipped with a five-speed forward, one reverse, transmission and a two-speed rear axle. The trucks weigh 5½ tons and carry a pay load of 8 tons.

Many special problems both of operation and maintenance arise due to the exceptionally cold winter conditions. An added factor is the frequent and excessive variation in temperature over the route, ranging as much as from −25° to −65° on a single trip. Added to the difficulties of keeping the vehicles in operation is the necessity of safeguarding the lives of any drivers who may be stranded in the colder weather.

This is taken care of by the convoy system. A convoy moves daily in each direction, made up of six to twelve, or more, trucks as required and shepherded by a 'wrecker'.

FREIGHT AND WAREHOUSING

All freight is received and checked at the Company's warehouse at Whitehorse. The old central warehouses at Mayo are no longer used, and all goods are delivered to the Elsa camp where ample warehousing capacity, both heated and otherwise, is available. Smaller warehouses are maintained at other operating camps.

POWER

Lack of adequate, inexpensive power has in the past been one of the chief factors retarding development of the district. With the exception of a small wood-burning plant put into operation in the early 1920's by Keno Hill, Limited, and which rapidly exhausted locally available fuel, all power generated to date has been by means of Diesel engines. Until recently, lack of adequate transportation facilities imposed a limit on the amount of fuel oil that could be imported, thus limiting power production possibilities and, in turn, mining operations as a whole. This situation has, however, been remedied by completion of the Whitehorse-Mayo road and United Keno Hill has since augmented Diesel capacity to a considerable extent.

All power generated is produced at a central plant, located at Elsa Camp, from four *VGB*-type Ruston-Hornsby engines with a combined rating of 2,310 b.h.p. Two older engines remain as stand-bys with combined rating of 480 b.h.p. Power is generated at 440V and stepped up to 2,300V and 6,900V, the latter for transmission to No Cash, Hector, and Calumet mines.

TABLE XI.— OPERATING COSTS, DIESEL GENERATING PLANT

Supervision	$0.001 per kw. hr.
Operating labour	0.003 "
Operating supplies	0.001 "
Repair labour	0.001 "
Repair supplies	0.006 "
Diesel oil	0.033 "
Lubricants	0.003 "
	$0.048 per kw. hr.

Direct operating costs of the power plant are shown in Table XI, the total being $0.048 per kw. hr. Fuel oil, the most expensive single item, costs $0.38 per gal. delivered at the power plant. The oil is delivered at Skagway in tankers and pumped over a pipeline to a tank farm at Whitehorse. Thence it is transported to Elsa in the Company's own tanker fleet of trucks. These trucks have specially constructed tanks, of rectangular shape, so that concentrates can be handled on top as a back haul. This experiment has worked out most successfully. Sufficient storage capacity is available at the mine to carry all requirements over the freeze-up and break-up seasons.

The necessity for the development of hydroelectric power has long been evident in the camp. In 1948, United Keno Hill Mines instituted surveys of local power sites which concluded in a recommendation for construction of a plant on the canyon of Mayo river, near its confluence with the Stewart. The Dominion Government became interested and, through the Northwest Territories Power Commission, carried out a survey in the summers of 1949 and 1950, with favourable results.

Late in 1950, an agreement was entered into between the Northwest Territories Power Commission and United Keno Hill Mines whereby the latter agreed to commit itself to the firm purchase of power at a minimum rate of 1,600 horsepower-years for twenty years, at a rate of $125 per horsepower-year. Thereupon, the Commission laid plans for construction of a plant and actual work was commenced in the spring of 1951. The work is progressing well and the scheduled date of completion, late in 1952, should be met without difficulty.

CAMPS

The operations of the Company already extend over a considerable area and are expected to expand and multiply to a marked extent in the future. Transportation between camps is at times a matter of some difficulty, particularly in the spring. Under the circumstances, the degree of centralization which should be employed is a matter which has been much debated but by no means fully solved.

In the matter of mechanical maintenance, for example, the policy is to maintain one central machine shop, with smaller shops as required at the various camps, capable of routine maintenance and repairs but not equipped to undertake the larger jobs. The same reasoning applies to other service departments.

With regard to personnel, the original policy of establishing bunkhouses and cookeries at each camp is gradually being shelved in favour of maintaining a few larger camps and moving the men, as required, by bus. This centralization is thought to be conducive to cheaper housing costs and better morale. It will also permit the Company to provide better recreational facilities.

At the present time, the Company maintains two main camps, at Elsa and at Hector, and is gradually building up a third at Keno City.

At Elsa camp are situated the mill, power house, central machine and electrical shops, laundry, and central accounting and engineering offices. Bunk-houses provide accommodation for 144 men and a staff house for 17 men. A new cafeteria-style mess-hall is under construction. Some twenty-five family dwellings are available and occupied.

The Hector camp is restricted to standard mine-head buildings. Bunkhouse and cook-house accommodation is provided for 208 men and a 24-man staff house is in course of

erection. There are four family dwellings, with more planned.

Both the Elsa and Calumet camps are heated through central steam, coal-fired systems. At Keno City, the Company has taken over three former hotels and is completely renovating and re-equipping these buildings, which will provide accommodation for from 40 to 60 men. Standard, modern bunk-houses, as at the other camps, will also be provided.

Smaller camps, with accommodation for from 10 to 25 men, are available at the Silver King, No Cash, Bermingham, Porcupine, and Ladue properties, and at the sawmill near Mayo.

Bunk-house and cookery operation result in a considerable loss to the Company, due to high cost of foodstuffs in the Yukon. Each employee is charged $2.25 per day for accommodation, while cost to the Company for the first half of 1951 averaged $3.02 per day.

Water Supply

Water supply, for fire-fighting and for domestic and industrial use, is a problem of some importance.

At the Elsa camp, water is obtained from four sources, namely: two creeks, one running through the camp and the other about 4,000 feet distant; drainage from the Elsa mine; and drainage from the Calumet mine, through a three-mile pipeline of 3-inch wood-stave pipe. The four sources tend to dry up in about the order named so that, in late February and early March, the mill may be seriously incommoded. Only the water from the two creeks is potable and, for a few weeks each year, it may be necessary to draw drinking water from as far distant as eighteen miles. To date, few of the houses are equipped with winter pipe-lines and water is delivered twice-weekly by tanker truck. As much water as possible is withdrawn from the mill tailings and recirculated through the circuit. Due to the presence of residual reagents in this water, it cannot be returned to the circuit indiscriminately and a dual circulation is therefore employed throughout the mill.

At the Hector mine, drainage water is picked up at the portal and pumped to the compressor cooling-water tank. Domestic supplies are drawn from this source, filtered and chlorinated, and circulated through the camp. The resulting product has tested satisfactorily both chemically and bacteriologically.

Both Elsa and Calumet camps are fully protected by sprinklers. At Elsa, the main storage is an underground sump, excavated on the 200 level of the mine, with a capacity of 100,000 gallons and a head of 175 feet above the portal. As the ground surrounding the sump is permanently frozen, a circulation of warm water is maintained by pumping 25 g.p.m. of Diesel-plant cooling water into the main discharge line and allowing it to overflow the sump dam. At Hector, the supply is a 75,000-gallon wood-stave tank, erected on the hill above the mine.

For future water supplies on the north side of Galena hill it will be necessary to go into the valley of the McQuesten river, which is broad and filled with numerous ponds. A survey to one of the larger ponds indicates a distance from the Elsa mill of three miles and a static head, against which it would be necessary to pump, of 650 feet. This will not be a difficult proposition once hydro-electric power is available.

Labour

The Keno Hill area is feeling the pinch of the current labour shortage to a considerable extent due to the relative remoteness of the camps, the lack of any large centre of population in the immediate neighbourhood, and the tradition of frontier-town irresponsibility, the flavour of which still lingers. For the past year it has not been too difficult to recruit men, but heavy turnover has posed a constant problem of training and supervision. Some relief has been obtained through employment of immigrant labour. Cost to the Company in 1951 of hiring services and paying fares will approximate $50,000.

Lumber and Timber

The Company owns its own sawmill on the Stewart river near Mayo, with timber limits some 25 miles above the town. However, due to the exhaustion of commercial stands below Fraser Falls on the Stewart, it is now necessary to move logging operations above that point to stands sufficient to carry normal operations for a number of years.

Apart from this, the Company has given financial support to two other sawmills situated near the Stewart Crossing on the Mayo-Whitehorse road.

A planing mill is operated at Elsa, where produce of any of the three mills can be dressed as required.

Mining timber of all sizes is at present purchased under contract, delivered to the mine. Costs per foot vary from 5¾ cents for small lagging to 25 cents for 10-inch top mine timber, all in 16-foot lengths.

Yukon Coal Company

Foreseeing the increasing price and scarcity of wood, United Keno Hill Mines in 1947 acquired control of the coal deposits on the Yukon river near Carmacks from the former operating Company. The Federal Government, with the object of securing a new and more abundant source of fuel for the Territory, agreed to advance funds for the rehabilitation and development of the mines, and production on a small scale was commenced in the fall of 1947. All necessary camp and mine buildings have now been erected, and active production is in progress. The coal is semi-bituminous, with a B.t.u. rating of about 12,000. It is friable and does not stand up well in lumps but can be burned in a wide variety of stokers with much success.

Over half the production is absorbed by United Keno Hill with most of the remainder going to Dawson and a limited tonnage to Whitehorse. Annual output is in the neighbourhood of 4,000 tons.

Acknowledgments

The writer wishes to express his appreciation to Mr. F. M. Connell, President of United Keno Hill Mines, Limited, for permission to publish this paper. He is also indebted to various department heads for much assistance in obtaining material and preparing drawings. It is fitting to remark at this point that without the intelligent and disinterested service of these same men over the past years, what is recorded above could not have been accomplished.

* * *

Discussion

Question: What is the average wage at Keno Hill?

Mr. Brodie Hicks: For surface men, $1.10 per hour; for miners, an average of $1.80 per hour, including bonus.

Question: Can you give some data on the Onek mine?

Mr. Brodie Hicks: There are two old shafts on the property, sunk by hand, some 600 feet apart, in both of which good ore was encountered. We are at present engaged in driving an adit to tap the ore below these shafts and, while we have encountered what we believe to be the vein and are obtaining some very good assays, it is still too early to assess the true value of the property.

An Electrolytic Etch Technique for Aluminum

And Its Application To Some Metallographic Problems

By E. C. PEARSON, G. MARCHAND,* and R. H. HAY**

(*Annual General Meeting, Quebec City, Que., April,* 1951)

(*Transactions, Volume LV,* 1952, pp. 348-354)

ABSTRACT

A polishing technique is described together with two anodic etching methods for aluminum and its alloys. Some of the optical and structural characteristics of the film formed by these etching methods are discussed and it is shown to what degree these properties of the film are related to the crystallographic orientation of the metallic substrate. Applications to the study of recrystallization and twinning phenomena and to the study of plastic deformation, banding, and texture estimation are discussed. Micrographs are shown to illustrate the manner in which the technique is particularly well suited to the study of such phenomena.

Introduction

The physical metallurgist is constantly faced with the problem of polishing and etching of metallic sections. Mechanical polishing is tedious, time consuming, and not applicable whenever a surface entirely free from plastic deformation is required. Etchants are generally required to reveal the grain structure in order to determine its size, shape, and regularity, or to bring up constituents. In the case of aluminum, the most common etchant to reveal grains is a mixture of three acids: HCl, HNO_3, and HF. This etchant is fairly satisfactory on certain alloys and for certain metallurgical conditions but in general fails to give a good definition of the grain on castings, on highly alloyed products, and whenever the grain is smaller than 100 microns in diameter. Besides size and shape, a grain, or crystal, has an orientation and may be endowed with several types of imperfections. Its orientation is of particular importance in the field of fabrication, where many problems arising in connection with forming, deep drawing, and finishing can be traced back to general or local textures (preferred orientation) of the grain. The present method consisting of electrolytic polishing and electrolytic etching followed by examination in polarized light will find an application in all these fields, and also should be useful to those working on problems such as recrystallization and plastic deformation. In a number of cases, it also proves to be a good supplementary method to standard X-ray and etch pit methods.

Polishing Technique

(*a*) A large number of electrolytes were tried before we chose the following one, which was found to give reproducible results:

	% by vol.
Orthophosphoric acid (85%)	70.0
Distilled water	2.5
Carbitol (diethylene glycol monoethyl ether	26.5
Hydrofluoric acid (48%)	1.0

In use, the composition of the bath changes fairly rapidly, resulting in a large decrease in conductivity. Even though it has to be replaced rather frequently, it lasts longer than most solutions which were tried.

Figure I is an exploded view of the set-up used. The metallic tank is coated inside and out with *Nukemite No.* 35; the cathodes are carbon, and the stirrer is of a conventional type with a plastic agitator. The technique used is essentially that described by Hone and Pearson (1) except that the electrolyte is maintained at 70°C. instead of 20°C.

(*b*) Rolled sheet and other 'as-fabricated' specimens having a good surface finish are currently electropolished without any prior mechanical polishing. If a greater degree of flatness is required, the samples are abraded down to 000 paper and in special cases polished on the first wheel with carborundum, diamond dust, or Brasso prior to electro-polishing. A characteristic of this method is the high current density required for polishing: 2 to 3 amp. per square inch. The voltage applied varies in our case from 30 volts in a fresh electrolyte up to 60 volts in an old bath. The electrolyte is generally replaced at the 50-volt point. The temperature of the bath is maintained around 70°C. in a thermostatically controlled tank. Strong agitation is provided to prevent local overheating and pitting attack of the sample. The whole polishing

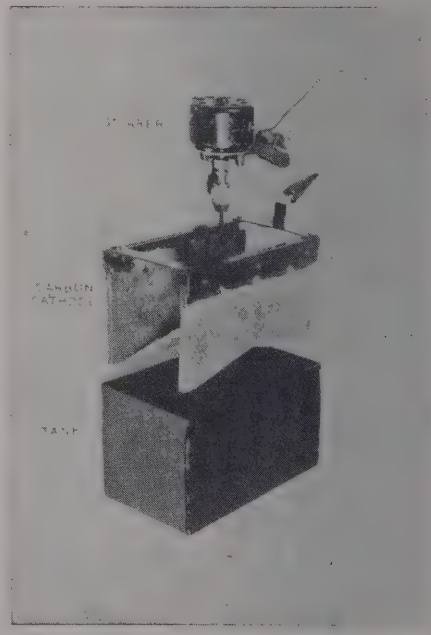

Figure 1.—Exploded view of electrolyte container used for polishing and etching. The holes and peculiar shape are to facilitate movement of electrolyte from tank wall to central portion. Chemical protection of aluminum tank is afforded by Nukemite No. 35.

*Aluminium Laboratories, Limited, Kingston, Ont.

(1) For references, see end of paper.

operation requires ½ to 2 minutes depending on the initial preparation, after which the sample is quickly washed in water and dried in a stream of air.

(*c*) The as-polished surface has very high specular reflectivity which does not appear to deteriorate after several months of exposure to the ambient atmosphere. This is probably because even the best polished surfaces obtained by this method are usually covered with a very thin film which is practically invisible under the microscope in ordinary light but may be perceived faintly under polarized light. This film may be dissolved in the *ASTM* oxide stripper (phosphoric-chromic acid mixture) but caution has to be exercised because in some cases evidence of attack resulting from this operation could be detected.

Etching Technique

The term etching as used here does not refer to the usual type of chemical attack but simply to an anodic treatment of the specimen as the result of which an oxide film is formed on the surface of the specimen under controlled conditions. The two methods we use for producing such a film are as follows:

In the first, the electrolyte is the same as the one mentioned above, but the temperature is kept at 20°C. The operation is started at a potential of a few volts which is increased gradually up to 50 volts. At this point the current density is about 100 milliamps per square inch. The voltage is kept constant for 20 minutes, which is the time usually required to form a surface film having optimum birefringence. Longer etching times will produce striations in the film.

Since these striations are useful in some cases, a second electrolyte was developed in which they form much more rapidly. The composition of this electrolyte is as follows:

Orthophosphoric acid (85%)	808 ml.
Carbitol	1,660 ml.
Hydrofluoric acid	320 ml.
Boric acid (anhydrous)	128 gr.
Oxalic acid (crystals)	72 gr.
Water (distilled)	1,012 ml.

This electrolyte gives the best results when operated at 35°C., using the following etching procedure. After immersion of the sample, the voltage is regulated to give an initial current density of about 5 amps per square inch for 30 seconds and then the voltage is lowered slowly until a current density of about ½ amp./sq. in. is obtained. This current is maintained for 1½ to 2 minutes. The sample is then rinsed in cold water and blown dry in a stream of air. The film produced according to this second method has a high birefringence and very well developed striations, but it rubs off easily and deteriorates in a few days.

Properties of the Film

Little is known about the *chemical* nature of this particular type of film; it is undoubtedly an oxide of aluminum but, as was found in the case of coatings produced in a sulphuric acid solution (2), it could be formed of more than one type of oxide, namely, amorphous oxide (anhydrous), boehmite (monohydrate), and perhaps a crystalline non-classical form of transition between these two.

It is generally considered that the substance of the coating itself is amorphous, which is difficult to reconcile with the optical activity of the film. K. Huber (3) and coworkers have tried to explain this property of the film in terms of the so-called "dispersed structure". This "dispersed structure" would arise from the formation of a system of pores in the film. Although both the oxide and the air in the pores are isotropic, the composite structure would behave optically as a birefringent crystal. This theory is supported by the fact that colour effects due to birefringence disappear when the film is impregnated with a liquid of high refractive index. The existence of pores can easily be checked by examining, under the electron microscope, an oxide film formed in the above electrolytes. A typical electron micrograph is shown in Figure 2. Such films used for electron microscope examination were made much thinner than those used in our current work so that they were transparent to electrons; they should, however, give us some indication of the structure of thicker films. The oxide in

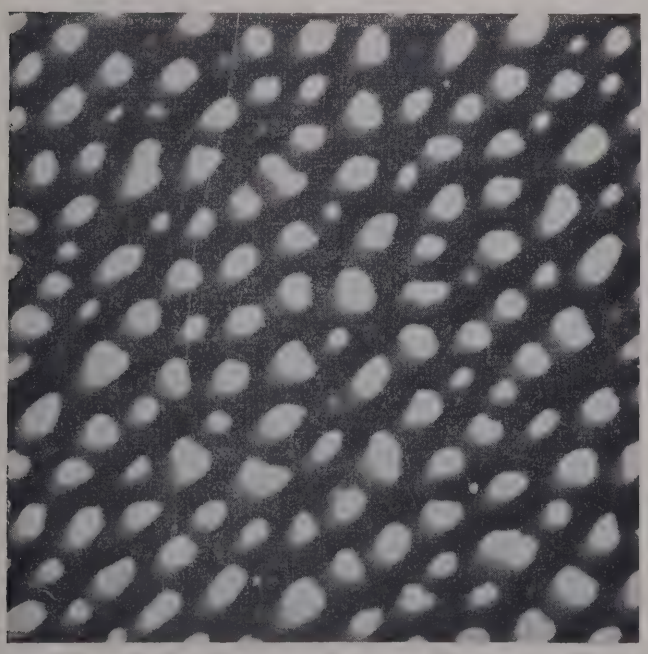

Figure 2.—Electron photomicrograph of pore structure in oxide film produced in first electrolyte. × 50,000.

Figure 3.—Electron photomicrograph of pore structure in film on planes close to 110 and 100. The pores are found to be much finer and more numerous on the 100 (upper half) face. Sample was etched in 47% HNO_3, 50% HCl, 30% HF, prior to anodizing in order to show orientation. Electrolytically etched in first solution for one minute. × 10,000.

Figure 4.—Electron photomicrograph of pore structure in film on planes close to the 100 and 110 planes. The pores seem to result from a double network of the oxide on the plane close to 100 and a difference in both size and shape of the pores is seen between the two grains. Sample was prepared by etching in 47% HNO_3, 50% HCl, 3% HF, and then electrolytic etching in the first solution for one minute. × 10,000.

Figure 5.—Electron photomicrograph of pore structure in oxide film in which the pores are lined up in rows parallel to the 100 direction. Etched in Tucker's reagent (45% HCl, 15% HNO_3, 15% HF) and electrolytically etched in first solution for one minute. × 10,000.

Figure 2 appears as a dark network, leaving a sizeable portion of the area unfilled. These unfilled portions will be referred to as pores. It has been claimed that the average size, density, and distribution of the pores is practically independent of the orientation of the substrate (underlying metal). The present observations did not support these views entirely. While in films formed on electrolytically polished surfaces only slight differences could be noticed, definite differences in the size, density, and distribution of the pores could be observed whenever such surfaces were attacked to develop etch pits, *i.e.*, in an acid mixture such as (HCl + HF + HNO_3). Hence, in Figure 3 the pores on the grain having a face close to one of the 100 planes (upper half) are appreciably finer and more numerous than those on one of the 110 faces of the other grain (lower half). The surfaces which are compared here are the flat areas between the etch pits; they are unattacked, or possibly evenly attacked, areas. In Figure 4 is shown another example of how the pore structure may vary with the change in orientation of the substrate; in this electron micrograph, the grain of the upper right, presenting a 100 face, seems to have a double network of oxide as if a fainter one was superimposed on the main one. In some cases, where the film is formed on an etched surface, a definite lining-up of the pores has been observed; on the electron micrograph shown in Figure 5 they appear to be in rows approximately parallel to the edge of the etch pits and in a 100 direction. The characteristic appearance of the pores seems thus to depend largely on the condition of the surface prior to the anodic treatment.

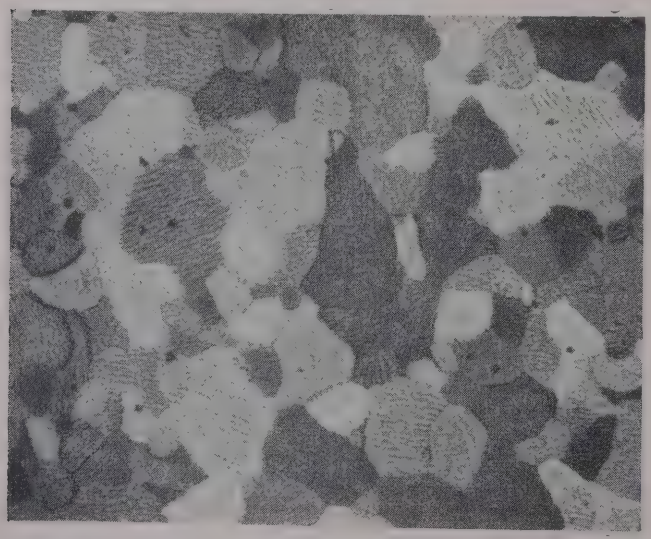

Figure 6.—Striations on randomly oriented, recrystallized, super-purity aluminum, electrolytically etched and photographed in polarized light with magenta tint insert (original in colour). × 200.

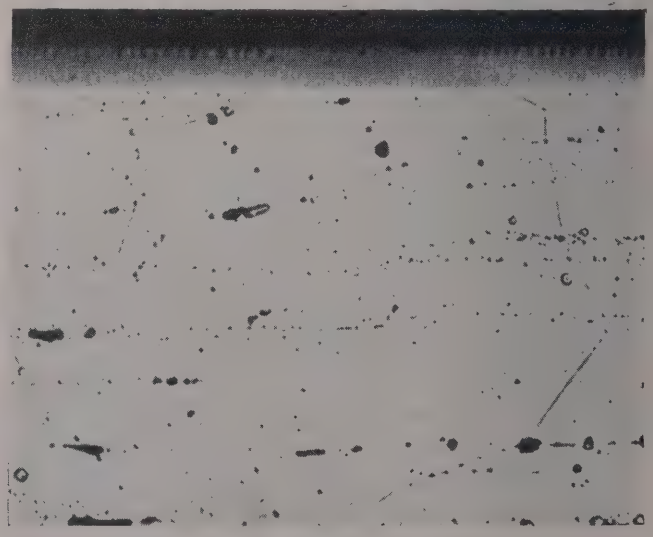

Figure 7.—Cross-section of sample having a film on which striations appear. The shape and depth of the striations are apparent. Sample was etched in diluted Tucker's reagent to show the grain boundaries in the aluminum. × 500.

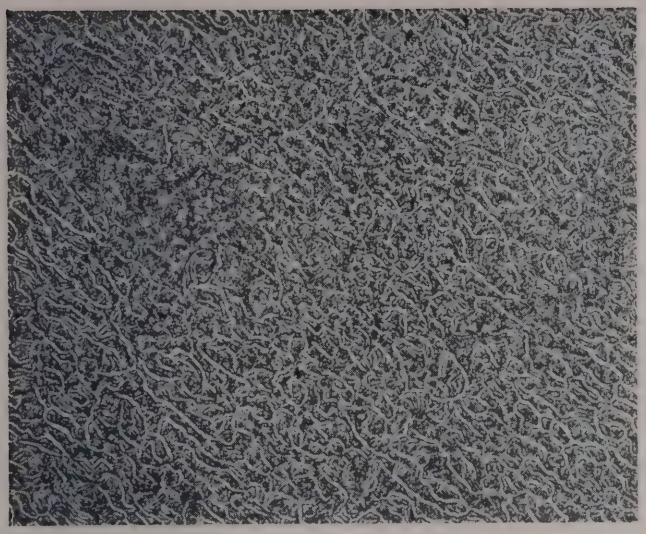

Figure 8.—Striations which appear on a plane close to the 100. The lack of directionality is apparent. Electrolytically etched in the second electrolyte. × 200.

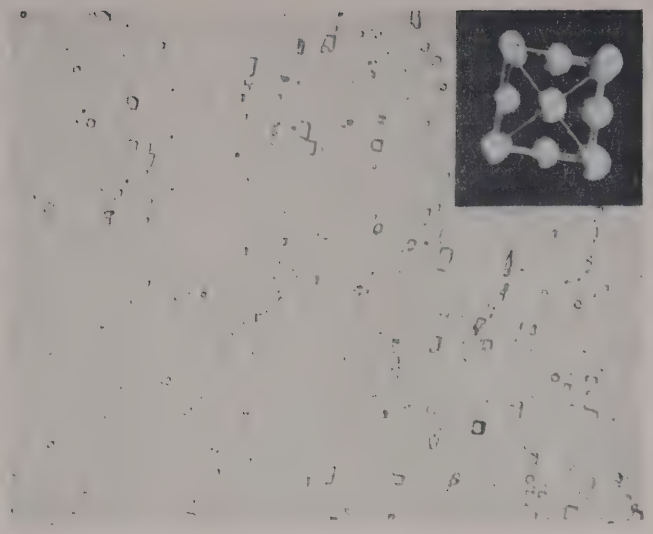

Figure 9.—Etch pits developed by etching the grain shown in Figure 8 in Tucker's reagent. The model of the atomic space lattice indicates the orientation of this grain. × 200.

An idea of what is meant by *striations* in the film may be acquired through Figure 6, which shows an electrolytically etched specimen of commercially pure aluminum. In each grain there is a set of approximately parallel lines and these lines have a different orientation from grain to grain. Figure 7 represents a cross-section through a striated film. The striations appear as furrows or grooves going about one-quarter of the way through the thickness of the film. These striations are possibly due to a selective dissolution effect occurring after prolonged etching.

When viewed in ordinary white light, the anodic film is colourless. In plane polarized light, *i.e.*, between crossed Nicols, a black and white effect is obtained. Some grains appear dark, some light, and upon rotation of the specimen each grain lights up and becomes extinct at specific angles. The anodic film thus behaves like an anisotropic, or birefringent, substance, *i.e.*, with two different indices of refraction in two directions. Since the amount of retardation introduced by the film is less than half a wave length, no interference colours are seen in white polarized light; instead, the grains vary from black to greyish-white of the first order; by the introduction of a magenta tint in the optical system, the total amount of retardation is increased by one wavelength and interference colours appear.

That the film bears a relationship to the substrate is obvious from the fact that, on recrystallized material, the colour is uniform over one crystal and changes abruptly at the boundary of two crystals having different orientations. When the stage is rotated, using the magenta tint insert, the colour of one crystal changes in 180° cycles. This property of the film is the same if it is detached from the base metal and examined in transmitted light.

The above discussion applies to anodic coatings formed on crystals presenting a face other than 100 or 111, for in these two special cases the film behaves like a uniaxial crystal with its axis perpendicular to the surface. It remains magenta over a 360° rotation, and without the magenta insert it remains extinguished. Thus it is not possible to distinguish optically between these two orientations of the crystal.

It must be remembered when examining these electrolytically etched specimens that grains of different colour have different orientations, but the converse is not necessarily true for the reason stated above and also because of the fact that two grains which are coloured the same for a given stage position may assume different hues upon rotation of the stage and hence have a different orientation.

Similar knowledge is provided by the coarse striations. Since they parallel the direction of vibration of the slow ray one can see at a glance the variation in extinction angle from grain to grain. The manner in which the striations are related to the substrate and also to the etching pits is illustrated in Figures 8 to 13. In Figure 8 it is seen that the striations travel at random; this surface corresponds to a 100 orientation, as seen in Figure 9, which shows some etch pits developed on the same crystal. A model of the unit cell indicates the orientation of this crystal, which was determined by X-rays. Figure 10 represents the striations as they appear on a 110 plane. By reference to Figure 11, it may be seen that the striations in Figure 10 run parallel to the trace of two 100 planes on the surface, *i.e.*, the ones making the smallest angle with the plane under observation; they also line-up with two sides of the etch pit. Two systems of striations often appear on some low-order crystallographic planes, as shown in Figure 12. By reference to Figure 13, it is seen that one of the systems appears to be parallel to the long side of the triangular etch pit; the other system is not obviously related to any important characteristic of the crystal.

Applications

(*a*) Macroetching to show up the grain structure is based in most cases on the production of crystal facets. The striations produced by electrolytic etching, having a characteristic orientation on each grain, serve the same purpose, and electrolytic etching has the added advantage of leaving most of the constituents substantially unattacked. It generally gives much better results than chemical etching on cold-worked and as-cast material. It fails in the case of alloys with a high silicon content. In alloys having appreciable amounts of copper, the etch will vary with the metallurgical condition. Figure 14 illustrates a section through a direct chill cast ingot as photographed in ordinary light. The various types of crystals are easily distinguished; namely, the columnar crystals formed near the

Figure 10.—Striations on a 110 face produced by electrolytically etching in the second electrolyte. × 200.

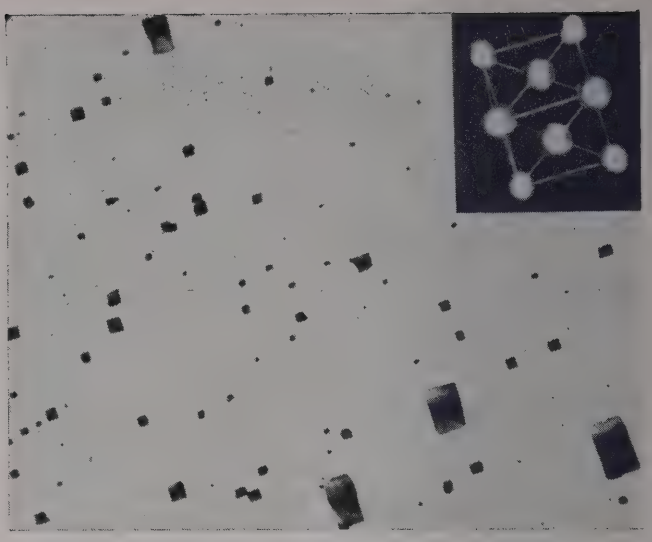

Figure 11.—Etch pits on grain seen in Figure 10; Tucker's etch. × 200.

Figure 12.—Double set of striations which appear on some low-order planes; electrolytically etched in the second electrolyte. × 200.

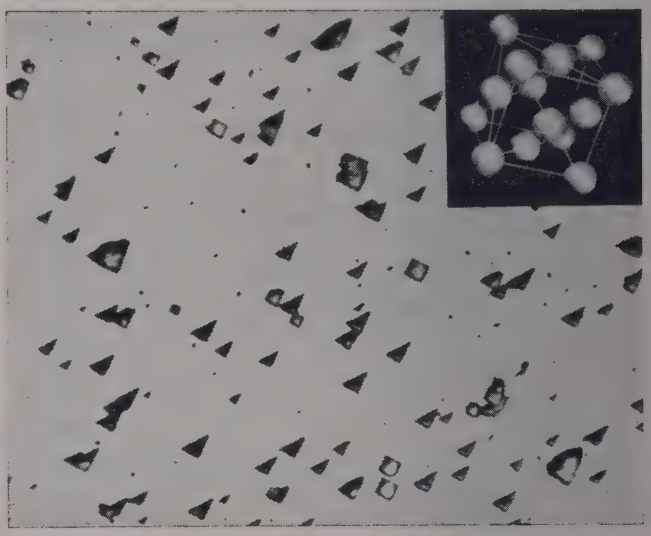

Figure 13.—Etch pits on grain seen in Figure 12; Tucker's etch. × 200.

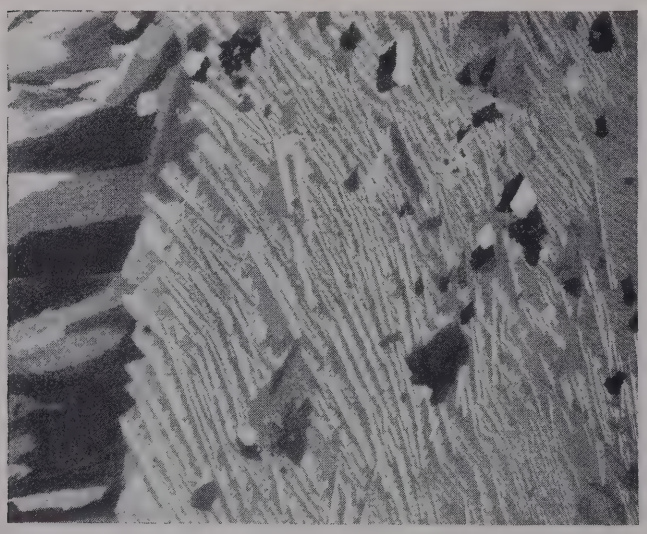

Figure 14.—Microscopic use of the electrolytic etch is illustrated by this cast ingot showing the 'feathery' structure. × 2.

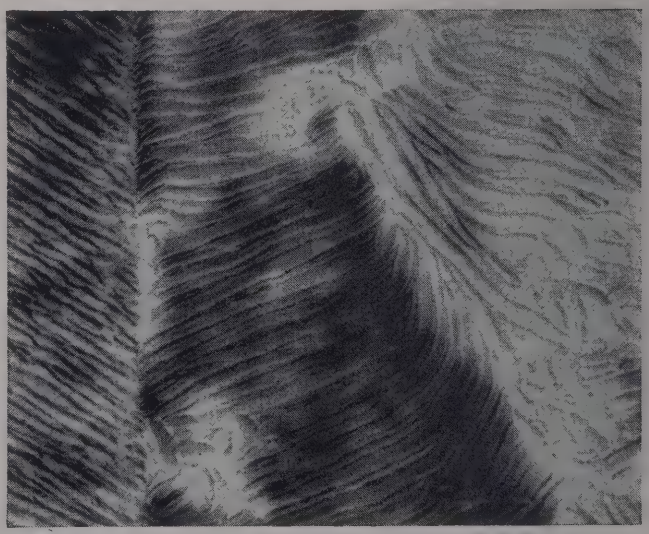

Figure 15.—Cold worked super-purity aluminum sheet. Differences in type of deformation may be seen between grain on the left and the remainder of the field. Electrolytically etched and photographed in polarized light with magenta tint insert (original in colour). × 500.

mould wall (left-hand side of the picture), then the so-called 'feathery' structure about which more will be said later, and finally, equiaxed crystals scattered through the 'feathery' crystals.

(*b*) The result of electrolytically etching a deformed structure is shown in Figure 15. The original print, taken in colour, revealed the variations of orientation of the substrate partly by changes in the type and direction of the striations and partly by differences in the colour. It is impossible to show here the gradual shifts of colour indicating the gradual change in orientation from one point to another, but the striations themselves reveal additional information. Depending on their original orientation and that of their neighbours, grains deform differently. In this photomicrograph of cold-rolled super-purity aluminnum, the crystal on the left appears to have been deformed by homogeneous gliding, for the striations in this area are fairly straight. The material in the rest of the area of this picture suffered considerable distortion and it is difficult to say whether we had two grains or one before deformation. A swirl-like pattern such as that found in the lower right corner of the photomicrograph may result from a more complex mode of deformation. The striations in this corner resemble those seen in Figure 8, which are found in 100 and 111 faces; however, in this case the colour indicated that this swirl-like pattern was not formed on one of these faces, for no part remained magenta on rotation of the stage.

(*c*) This technique has been used to some advantage in the study of the formation of texture and also in the field of recrystallization. For instance, Sperry (4) has developed a simple method for following grain boundary migration in aluminum based on successive annealing and electro-etching treatments. The intermediate positions of the boundary are located by the effect produced by the previous etch. The final position is revealed by the colour effect of the film.

Figure 16 shows a high-purity aluminum alloy containing 0.11 per cent Si, cold rolled 75 per cent, and then annealed thirty minutes at 320°C. This sample was partially recrystallized and shows that recrystallization does not occur at random in all the deformed crystals but that certain grains recrystallize earlier than others. In the original micrograph in colour, the unrecrystallized material was easily distinguished by its larger size and its peculiar varieties of colouring from area to area. On the right-hand side a region may be observed where almost complete recrystallization has occurred. It was noticed on the original photo that the new grains had a predominant colour (red in this instance and dark in the monochrome print). At higher magnification, all the striations are clearly seen to be parallel. Presumably these grains have a common preferred orientation. Beck (5) has used this technique to illustrate the phenomenon of preservation of a deformation texture after annealing at low temperature. In partly recrystallized samples such as we have here, there is a predominance of two colours in the deformed material; these two colours correspond to the two components of the classical rolling texture of face-centred cubic metals. They may be distinguished one from the other by calling them 'red' and 'green'. In this reproduction, the original 'red' areas are dark and the 'green' areas are light. Upon annealing, the 'green' deformed areas tend to give rise to 'red' recrystallized grains, and *vice versa*, so that the rolling texture is substantially preserved.

The central banded area of Figure 16 has been enlarged in Figure 17 to reveal the striations. They appear to form a herring-bone pattern and constitute a further illustration of the formation in one crystal of the duplex elements of the rolling texture just mentioned.

(*d*) Another specific problem to which electrolytic etching was applied is that of 'banding' in rolled sheet. Banding is a defect which results in streaks of different shades on etched or anodized sheet. It may be due to an uneven distribution of insoluble constituents or to local textures. Figure 18 shows a case of 'banding' which was due to the latter cause, namely, streaks of grains having a preferred orientation. This constitutes one means of distinguishing between the two types of banding.

(*e*) Although aluminum has a face-centred cubic lattice like copper, nickel, and silver, it is generally considered as a metal in which

Figure 16.—Partially recrystallized super-purity aluminum with 0.11% silicon added. In some regions, a cold-worked structure still prevails whereas in other regions recrystallization has started. Of particular interest are the parallel bands in a single deformed grain and the similarity of orientation of a group of non-recrystallized grains in a single parent grain. Electrolytically etched and photographed in polarized light with magenta tint insert (original in colour) × 100.

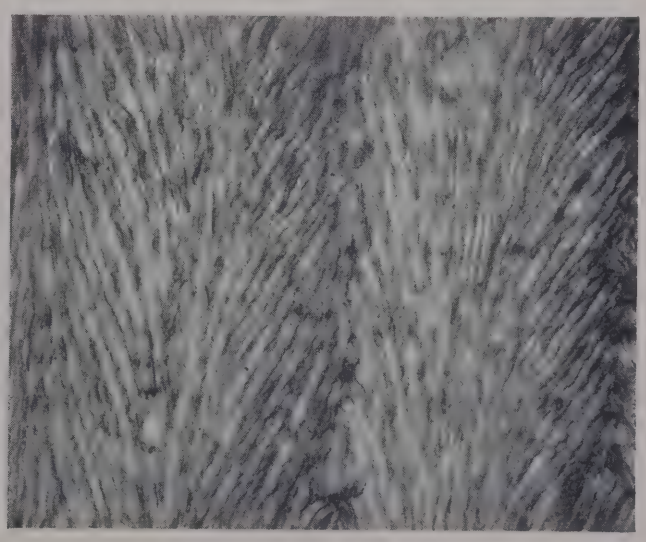

Figure 17.—Detail at a higher magnification of the parallel bands in middle of Figure 16. From the striations, it may be seen that adjacent bands have widely different orientations and that within one band and near the band boundaries there is a gradual change of orientation. Parallel portions of the grain have rotated in a manner suggestive of twinning. Electrolytically etched and photographed in polarized light with magenta tint insert. × 500.

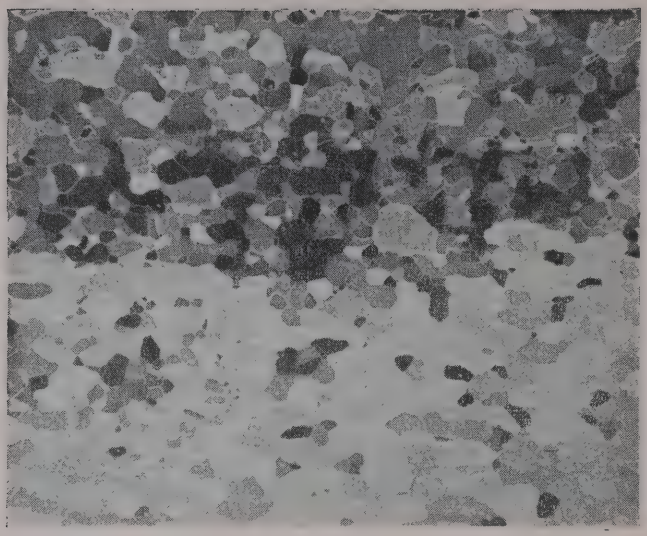

Figure 18.—Strong localized perferred orientation, which results in a banded structure. Grains in top half of field have a similar orientation but differ in orientation from those in bottom half. Electrolytically etched and photographed in polarized light with magenta tint insert (original in colour). × 75.

Figure 19.—Annealing twin in super-purity sheet with 0.04% Mn added. The left- and right-hand portions of the large central grain were found to have identical optical characteristics. Electrolytically etched and photographed in polarized light with magenta tint insert (original in colour) × 75.

annealing twins cannot form. Actually, annealing twins are present in recrystallized aluminum as in all other face-centred cubic metals. Their frequency of occurrence is rather low and they can be easily missed if the section is etched in the conventional way. Figure 19 is an illustration of a classical type of twin formed in high purity aluminum.

Another form of twin, shown in Figure 20, is frequently encountered in chill-cast ingots and in weld beads of aluminum. These may be referred to as 'solidification' or 'dendritic' twins, and they constitute what has been previously called 'feathery' structure. Lacombe and Hérenguel (6, 7) have shown that they are twinned lamellae with their edge in the direction of heat flow. The twin boundary is a plane of the 111 family; two systems of twins may grow together, as in the case illustrated. Although this structure represents a very high degree of preferred orientation in a cast ingot, it was not found to lead to particularly pronounced preferred orientation in wrought commodities produced from it.

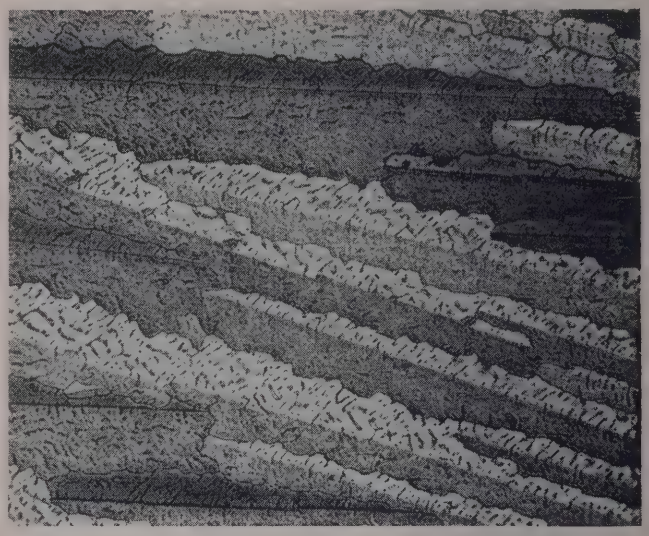

Figure 20.—Chill cast Alcan 1S alloy showing dendritic type of twinning. Alternate areas had the same optical characteristics. Electrolytically etched and photographed in polarized light with magenta tint insert (original in colour) × 50.

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Potential Sources of Alumina in Canada

By W. K. GUMMER *

(*Transactions, Volume LV*, 1952, pp. 355-360)

Introduction

THE LOCATION of Canada's present aluminum industry is determined by availability of hydro-electric power, and ready access to ocean transport facilities. The chief raw material for the production of the metal is imported by ship over several thousand miles of water. In times of freedom of shipping and uncontested movement this provides the industry with its cheapest method of operation. However, during the early part of the last decade the general conditions of ocean and rail transportation were such that interest in the treatment of materials other than bauxite was greatly stimulated. This was the case not only in Canada, of course, but in most other countries.

Canada has no deposits of bauxite, and much research was carried out on other aluminous materials, chiefly silicates, in an effort to develop more or less economic and feasible methods of treatment. In the United States, this research was considerably more extensive. Some projects were early shown to be of little or no value, whereas others, including the widely publicized lime-sinter and soda-lime-sinter methods, became important. In North America, however, apart from the application of the sinter method to the hitherto waste-product (red mud) of the normal Bayer plant operation, none of these processes contributed appreciably to the production of aluminum during the war. In certain European countries, on the other hand, largely because of their situation with regard to bauxite deposits, they had a more general application, and Germany, for example, produced alumina from clay, ashes, and slag by at least four methods.

It is quite conceivable that, had Canadian bauxite imports been still more seriously curtailed, our investigations and those of others would have led to actual large-scale production of alumina from siliceous ores. This situation nearly materialized in Canada, but bauxite supplies were maintained aud bauxite remains today the sole source of alumina even though imported from British Guiana and other places as far or farther from the Arvida Works.

At one time during the last war, Aluminium Limited and its affiliates fielded a large group of geologists, mineralogists, engineers, and chemists, who applied their combined efforts to new and untried sources of the metal. This paper is intended to provide a compilation of data obtained by field men and the various laboratories on rocks collected from Cape Breton to British Columbia. It includes also much information from government reports, particularly those on clays and shales and written from the point of view of the clay-consuming industries. This information is here assembled with a different approach, that of the alumina content and the value of the rocks as what we may term protores of aluminum.

Aluminum in the Earth's Crust

Aluminum is estimated to comprise slightly more than 8 per cent of the earth's solid crust to a depth of ten miles. Averages of large numbers of igneous and sedimentary rock analyses show that both the 'average' igneous rock and the 'average' shale contain just over 15 per cent Al_2O_3. This is worth emphasis, as it means that only the unusual rock may prove of interest to the aluminum industry, certainly for a long time to come. Bauxite generally contains 50 per cent or more alumina that is extractable by Bayer treatment. Materials that during the war years were considered as substitute sources include bauxitic and kaolinite clays with up to 40 per cent alumina, anorthosite with nearly 30 per cent, and nepheline syenite containing 25 to 30 per cent alumina and also an attractive amount of alkalies.

In absolute amount, such alumina-rich materials are of very minor importance compared with what may be termed run-of-the-ground clay and granite with less than 20 per cent alumina. It is also apparent that even the 'high-grade' rocks are far removed in alumina content from the marginal bauxites that are not used at present, and these latter would certainly be preferable for industry as long as they can be imported with impunity. If they cannot be obtained, then the industry is faced with a decrease in ore grade by a factor of approximately 2. This implies that, short of major improvements in the economics of treating the silicate materials, the aluminum industry in Canada will operate on low-grade bauxites rather than on the silicate rocks.

The Canadian Picture

Preferred Materials

In this section we include rock decomposition products in general. From our point of view, the most important of the materials of this class are naturally aluminous laterites and bauxites. The known extensive deposits of bauxites amenable to the Bayer process as applied on this continent, with only a few important exceptions, are found in tropical and sub-tropical countries. They result from a particularly complete type of weathering, and similar grades of ore may be formed from quite different source rocks. The process of bauxitization is believed to be going on at the present time in more than one area. The concensus of opinion — and this is not new — is that in Canada, which recently in the geological sense was still further removed from tropical climates than it is today, extensive young bauxites are not to be expected. We know that, in bygone eras, tropical climates must have prevailed over large areas in Canada, but to date no 'fossil' bauxite deposits have been located. Buried bauxites are known in other parts of the world, but in Canada we seem to have missed out. If bauxites formed here to any extent, they must largely have been removed. Such deposits are not

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physically resistant, and certainly the action of glaciation would be adequate to destroy them and distribute the materials elsewhere in unrecognizable form.

This probability of removal of 'soft' deposits applies equally as well to kaolinite as to bauxite. However, in the case of kaolinite, there appear to be a few exceptions, some even in the Precambrian shield area.

Laterite

Laterite occurs near Sooke inlet, on Vancouver island; it is high in iron, and very siliceous, containing less than 35 per cent alumina. Some experimental work has been carried out by others on this material. Still lower grade lateritic deposits occur at Stony Creek on Vancouver island, and on other nearby islands.

Mention may here be made of small and local deposits of laterite-like material existing here and there throughout Canada, some containing more than 20 per cent alumina. It is a stretch of the imagination to consider these as incipient bauxites although they appear to consist of hydrated iron oxides, clay minerals, and residual feldspar from which the clay mineral is developing. Some at least of the small samples that we have had occasion to examine have probably come from the oxidation zone of minor sulphide deposits.

Residual Clay

For the same reason that deposits of bauxite or laterite of great extent are not expected in Canada, deposits of residual kaolinite are rare; but some are known and one at least has been worked. At St. Remi d'Amherst, Quebec, occurs a large body of quartz and kaolinite as an apparent replacement development in Precambrian quartzites (1). It is discussed here because, irrespective of origin, it is a 'soft' deposit and its occurrence in a glaciated region has been described as "remarkable". A limited amount of high-grade kaolinite was found, and lower grade material is amenable to beneficiation by simple washing. Although this deposit has furnished the best grade kaolinite to come to our attention, it does not offer any appreciable tonnage as a source of aluminum should such material become important.

Elsewhere are deposits of similar mineral constitution. Material near St. Jovite is composed of kaolinite and abundant fresh quartz grains, most likely the result of the alteration of all minerals but quartz from a granitic source rock. Kaolinized feldspathic material from the Thetford area in Quebec is very similar except that it contains several per cent of titania, concentrated during the kaolinization process.

Small deposits of kaolinite are reported in several places in the Prairie Provinces, such as Moose lake, and Punk island in lake Winnipeg, Manitoba (2), and near the Wappeweka lakes, Saskatchewan. Secondary kaolinite occurs at Giscombe rapids, north of Prince George, British Columbia. This is one of the larger deposits; it is 20 to 30 feet thick and is of considerable length, with a grade of 28 to 30 per cent alumina. Residual clay is also reported at Williams lake, Vancouver island.

To conclude this section, it may be said that Canada offers little hope for the discovery of widespread bauxitic or even high-grade kaolinite deposits. What does occur is low grade or limited in quantity, or both. If the worst happens and the required raw materials cannot be imported, other materials will be needed to maintain the industry. The succeeding parts of this paper discuss the possible domestic substitutes that may be located.

TABLE I.— ANALYSES OF LATERITES AND KAOLINITES*

	I	II	III	IV	V	VI
L.O.I.	n.d†	11.9	10.5	13.4	7.4	8.6
SiO_2	27.0	31.7	32.3	46.1	68.9	63.3
TiO_2	n.d.	2.9	2.1	n.d.	n.d.	n.d.
Al_2O_3	34.9	23.8	26.6	39.5	21.9	24.8
Fe_2O_3	20.6	27.6	25.5		n.d.	1.1
MgO	n.d.	n.d.			n.d.	0.9
CaO	n.d.	n.d.		1.0	n.d.	tr.
Alk.	n.d.	n.d.			n.d.	1.5

I — Laterite, Sooke inlet, Vancouver island.
II — Laterite, Stony creek, Vancouver island.
III— Laterite, Texada island.
IV— Kaolinite, St. Remi d'Amherst (washed product) (1).
V — Kaolonite and quartz, St. Jovite, Quebec.
VI— Residual clay, Punk island, lake Winnipeg. (2).

*Unless otherwise credited, analyses given in this paper were made by the Aluminum Company of Canada, Limited, Arvida.
†Throughout the various tables of analyses, "n.d." means "not determined".
(1), (2) For references see end of paper.

TABLE II.— ALUMINA-RICH MINERALS IN CANADA

NAME	APPROX. % Al_2O_3	NAME	APPROX. % Al_2O_3
Albite	19	Kaolonite	40
Aluminite	30	Kyanite	63
Alunite	37	Labradorite	30
Andalusite	63	Muscovite	39
Andesine	27	Nepheline	33
Anorthite	37	Orthoclase	18
Beryl	19	Pyrophyllite	28
Biotite	24 (var.)	Sillimanite	63
Corundum	95-100	Sodalite	32
Dumortierite	65	Spinel	72
Epidote	24	Staurolite	56
Garnet	20-25	Topaz	55
Halloysite	37	Tourmaline	30
		Zoisite	34

Aluminum-Rich Minerals (Other Than Hydroxides)

In Table II is given the alumina content of some twenty-five more or less common minerals which are common or are known to occur in Canada and which, under certain conditions, may be sufficiently concentrated that their occurrence is unusual and of commercial interest.

There are many other minerals containing from 10 to 20 per cent alumina; for example, several sulphates in this range have been found in isolated or minor deposits. Of the minerals listed in the Table, the following have been or are being investigated, or actually employed, as sources of aluminum in

Canada or elsewhere: andalusite, plagioclase feldspar as in anorthosite, nepheline rocks, alunite, and kaolinite.

The alumina percentages refer, of course, to the pure minerals. Rocks of real monomineralic nature are unfortunately rare, but examples include anorthosite and kaolinite, and in these the alumina may approach closely the theoretical figures given above. Not all the minerals could be treated by any one process, because of differences in chemical composition, refractoriness, and so forth, and none of them are amenable to the Bayer process.

Silicates of Igneous Origin

The alumina contents of six classes of igneous rocks may be listed as in Table III.

Those rich in alumina are seen to be the unusual types. The known area of quartz-free alkaline rocks comprises only a fraction of a percent of the total known area of igneous rocks. Anorthosites are somewhat more abundant.

Nepheline Rocks

Of the syenitic rocks, those containing nepheline have been investigated in detail as sources of aluminum. In Canada, the American Nepheline Corporation and the Aluminum Company of Canada, Limited, co-operated early in the last war in research on nepheline syenite. This showed that, although alumina of satisfactory purity can be obtained by the lime-soda-sinter method, the cost of this alumina greatly exceeded that of Bayer process alumina.

Canada has several localities noted for alkaline rocks. Of these the best known are perhaps those in the Bancroft-Gooderham and the Blue Mountain areas of Ontario. There are less extensive and less well-known bodies of similar type near Port Coldwell, and in the French River and Rainy Lake areas, in Ontario, Quebec, British Columbia, and the Northwest Territories also have nepheline rocks.

The minerals in nepheline rocks include nepheline itself — its presence is responsible for the unusual alumina content — alkaline feldspars, ferromagnesian minerals, and iron oxides. Corundum may also be present, as at Craigmont in particular, where the deposits have been worked for their corundum content. It appears to be a common characteristic of these rocks, particularly perhaps of those of the Bancroft and the Port Coldwell regions, that they

Table III.— Alumina in Igneous Rocks

Class of Rock	% Al_2O_3
Rhyolite and granite	11—16
Syenite and nepheline syenite	16—30
Diorite and monzonite	14—18
Gabbro and norite	12—21
Anorthosite	20—34
Pyroxenite, peridotite	2—9

are banded or zoned, and vary rapidly in composition over short distances. For this reason, the quarrying of nepheline rock of consistently suitable grade might prove difficult, and would necessarily be of a selective nature.

The greatest amount of study has been carried out in the Bancroft area, where a part of the overall belt of alkaline rocks was explored by detailed geological surveys and diamond drilling in 1941 (3). Just east of the village of Bancroft itself, a great tonnage of rock averaging 24 per cent alumina has been shown to exist. Of many millions of tons, only a relatively few thousand approach 30 per cent alumina, and that in separate bodies. Laboratory investigations showed that such rock is readily handled by the sinter treatment.

Nepheline rock of very similar nature, averaging about 24 per cent alumina, is actively quarried at Blue Mountain, near Peterborough, for ceramic purposes.

The rocks of the Port Coldwell area are of lower grade, as are those of the other areas referred to.

Representative analyses are given in Table IV.

Anorthosites

Anorthosites are by definition rocks composed essentially of lime-soda feldspars. The most preferable rock as a source of alumina would, of course, be one containing only the anorthite molecule, but such rocks either do not exist or are extremely rare. Typically, anorthosites are coarse-grained, of rather steady composition over large extents, and, in addition to andesine or labradorite feldspar, contain minor amounts of the minerals characteristic of the rocks of the gabbro class. Gabbro is frequently found in intimate association with anorthosite bodies.

The Quebec anorthosites, exposed over an area of several thousand square miles, are formed chiefly of a feldspar with about 50 per cent of the anorthite molecule. The rock at Arvida contains in the neighbourhood of 28 per cent alumina, and similar rock occurs in an oval-shaped area (roughly 80 by 150 miles) that extends to the north and east from lake St. John. Anorthosite has been sampled at lake Pipmaukin, about 90 miles northeast of Arvida.

Indeed, the Province of Quebec is particularly well endowed with anorthosite, now exposed in the roots of the ancient Laurentian mountains. A series of anorthosite bodies is known to occur along the north shore of the St. Lawrence river from just north of Montreal to the extensive exposures in which the Allard Lake titanium ores are found. The possibilities of finding additional occurrences are great.

Table IV.— Analyses of Nepheline Rocks

	I	II	III	IV	V
L.O.I.	2.8	n.d.	n.d.	0.9	0.8
SiO_2	47.9	42.8	58.3	56.0	53.4
TiO_2	0.2	n.d.	n.d.	0.2	0.6
Al_2O_3	25.0	26.9	23.8	22.0	21.0
Fe_2O_3	4.8	n.d.	1.9	3.5	1.7
CaO	3.1	n.d.	1.9	n.d.	2.9

I — Average of 1,330 feet of drill core, Bancroft area. Average Al_2O_3 for all drilling samples = 23.8%. MgO was less than 0.5% in most samples.
II — Average of 100 feet of drill core, selected, Bancroft area
III— Blue Mountain, Ontario. Surface sample.
IV— Port Coldwell, Ontario. Surface sample.
V — Nepheline syenite, Ice River, British Columbia (4).

A rock containing nearly pure anorthite has been described from Yamaska mountain, but it is not monomineralic. Some of the rock forming Brome mountain contains more than 27 per cent alumina, but such rock is of limited extent.

Anorthosite from near Minden, Ontario — adjacent to the nepheline rock belt — has been sampled and has been found to be very similar in composition to the Quebec anorthosites. Anorthosite from Parry Sound has been reported to contain as much as 31 per cent alumina (5), although our analyses indicated a lower figure, 28 per cent. Anorthosite at Sooke, Vancouver island, is reported to contain, by volume, 90 per cent of a feldspar of composition about Ab_2An_8 and therefore its content of alumina might approach 32 to 33 per cent. This rock grades into gabbro.

Should an economic and competitive process be realized for the large-scale extraction of alumina from anorthosite for metal production, astronomically large tonnages of the 'ore' would be available. Laboratory studies have shown that the Shipsaw anorthosite is amenable to the sinter treatment.

Representative analyses of anorthosites are given in Table V.

TABLE V.— ANALYSES OF ANORTHOSITES

	I	II	III	IV	V
L.O.I.	n.d.	0.3	n.d.	0.8	n.d.
SiO_2	53.0	53.0	52.1	51.1	52.9
TiO_2	0.2	0.2	0.3	n.d.	0.2
Al_2O_3	29.6	27.4	27.0	30.5	28.1
Fe_2O_3	0.7	2.5	2.6	n.d.	1.5
MgO	n.d.	1.3	n.d.	tr.	1.5
CaO	11.8	10.4	10.4	12.4	10.6
Na_2O	4.1	4.9	n.d.	4.3	5.0
K_2O	n.d.	n.d.	n.d.		

I — Selected andesine-labradorite, Shipshaw area.
II — Surface sample, Shipshaw anorthosite.
III— Average of six drill holes (1,129 feet), Shipshaw area.
IV— Anorthosite (scapolite-bearing), Minden, Ontario.
V — Anorthosite, Parry Sound area, Ontario.

Miscellaneous Igneous Materials

In addition to the rock species mentioned above, certain others contain interesting quantities of alumina. Generally, however, they are also rich in ferromagnesian silicates, or have low alkalies (which are advantageaous in any sintering process), or are curiosities rather than extensive occurrences.

As already mentioned, gabbros may contain up to 25 per cent or more alumina by virtue of their relation to anorthosites; however, the purer feldspathic rocks would be preferable. Quartz-feldspar bodies are abundant throughout Canada and many have supplied ceramic industries for years. In these, the feldspars are commonly potash and soda varieties, occurring together. Thus, the alumina content of such rocks tends to be below 20 per cent and the silica content well above 60 per cent.

Nepheline pegmatites are prominent in the Gooderham area; they contain chiefly albite and nepheline and their alumina content commonly exceeds 25 per cent. The bodies are limited in extent, and the distribution of the component minerals is rather sporadic, with albite generally predominating. The corundum-bearing pegmatites and gneisses of the same area are very minor in volume and may be ignored.

Sedimentary Clays and Shales

Alumina-bearing sedimentary rocks, consolidated or otherwise, are the most extensive, on the surface, of all materials that we might consider protores of aluminum. For this reason, and also because several other countries were considering them as sources of the metal, we undertook in the early years of the past decade an intensive programme of sampling across Canada. The results of this investigation, taken in conjunction with published analyses, have shown that the bulk of Canadian clays and shales — from Palaeozoic to Recent in age — are low grade in comparison with the other protores of aluminum.

Some general statements may be made regarding these rocks, before citing examples. For instance, the glacial and post-glacial clays, in particular, are characterized by admixture of sandy and silty components that lead to high silica content and therefore uninteresting alumina content. Some relatively good grade clays are associated with coal seams, but are generally in thin rather than thick beds. It is not difficult to locate clays with upwards of 30 per cent alumina, but it is difficult to find such deposits of worthwhile thickness or where many feet of useless material does not overlie them, thus detracting greatly from their importance.

Many rocks are termed clays without much thought to their actual mineral make-up, but the rock that will interest an aluminum industry first is the one that contains kaolinite, and the more the better. A good brick clay, and even a good refractory clay, is not necessarily a good aluminum ore.

Clays that are known from our surveys and from the literature to contain 30 per cent or more alumina are found in Nova Scotia, Quebec, Ontario, Saskatchewan, and British Columbia. In our records, the deposits of none of these exceed 10 feet in vertical extent. The best from all aspects are the clays of Musquodoboit, Nova Scotia; of the Moose and Mattagami Rivers area, northern Ontario; of the Willows area, Saskatchewan; and in the Fraser River valley, British Columbia. These last appear to be far and away the most interesting from the points of view of grade, extent, exposure, and access. But their overall value does not add up to their being the next raw material for production of aluminum.

Every province can provide some material with around 25 per cent alumina, and in many areas deposits of this type are widespread. For example, of about forty samples from Nova Scotia, approximately one-fifth contained between 23 and 27 per cent alumina; none of these samples represented more than 10 feet thickness. The Saskatchewan occurrences are much better in this regard; four samples in this grade-range from the Willows area average 16 feet thickness.

Table VI presents averages and selected analyses of clays across Canada.

TABLE VI.— REPRESENTATIVE ANALYSES OF CLAYS

	I	II	III	IV	V	VI	VII	VIII	IX	X	XI	XII	XIII	XIV	XV
L.O.I.	8	7.2	8.2	10.8	9.9	4.8	5.6	3.8	13.6	5.7	8.7	23.9	10.1	7.0	4.6
SiO_2	58	53.7	56.7	54.2	53.4	56.0	54.4	59.5	51.4	57.9	55.9	51.2	60.0	66.3	54.2
TiO_2	1	n.d.	0.9	1.3	n.d.	n.d.	n.d.	n.d.	n.d.	n.d.	n.d.	n.d.	0.8	0.6	0.6
Al_2O_3	18	22.4	20.8	29.7	25.0	17.3	19.2	17.5	31.1	20.0	18.5	24.0	25.1	19.4	21.9
Fe_2O_3	7	n.d.	5.6	2.3	n.d.	n.d.	7.3	7.4	1.4	6.8	6.6	tr.	1.6	2.9	8.0
MgO	n.d.	n.d.	n.d.	n.d.	n.d.	n.d.	3.7	3.3	0.1	3.3	2.7	0.3	n.d.	n.d.	n.d.
CaO	4	0.7	2.2	0.3	0.8	6.0	4.6	2.6	0.4	2.9	3.8	0.6	0.2	0.6	1.0
Alk.	n.d.	n.d.	n.d.	n.d.	n.d.	n.d.	5.0	4.6	tr.	n.d.	4.1	n.d.	n.d.	n.d.	n.d.

I — Average of 30 New Brunswick clay samples.
II — "Coal rock", Minto, New Brunswick (8.5 ft.).
III — Average of 36 Nova Scotia clay samples.
IV — Clay, Middle Musquodoboit, Nova Scotia (6 ft.)
V — Average of 5 clay samples, Sidney area.
VI — Average of 33 samples, Arvida area, Quebec.
VII — Champlain clay, Renfrew, Ontario (6).
VIII— Post-glacial clay, Kingston, Ontario (6).
IX — Best of several Cretaceous clays, James Bay basin (6).
X — Lorraine shale, Toronto (6).
XI — Queenston shale, Beamsville, Ontario (6).
XII — Bentonite, Morden, Manitoba (7).
XIII— Average of 9 samples, Willows area, Saskatchewan.
XIV— Average of 8 samples, East End area, Saskatchewan.
XV — Slate, Slate mountain, Alberta.

Other Materials

Certain metamorphic rocks derived from alumina-rich shales and clays are rich in alumina. Such rocks are not rare among the old sediments of the Precambrian.

Andalusite, staurolite, and muscovite are minerals found in schistose rocks that may contain a good deal of alumina. An example of the first is included in Table VII. Mica schists will contain alkalies (chiefly potash) in addition to alumina. Very pure muscovite schists are known in several localities; some are rather inaccessible, others are more readily visited. Near Sioux Lookout, Ont., several bands of such schist have been exposed and samples of some of them have been analyzed at Arvida (*see* Table VII). However, it is unlikely that any of these schists are to be found in sufficient quantity to raise real interest.

A curious rock from near Sharbot lake, southeastern Ontario, was found to contain upwards of 30 per cent alumina, accounted for by a small amount of corundum and an aluminous amphibole.

Pyrophyllite, a mineral related to the clay minerals and outwardly resembling muscovite, occurs in many places in Canada. At Kyoquot sound, Vancouver island, there are reported to be a million tons running 10 to 25 per cent alumina, with 72 to 88 per cent silica. The host rocks are quartz-sericite alunite schists, which themselves are of interest since the alunite, of the sodic variety, forms from 20 to 45 per cent of certain zones (8). Several hundred thousand tons have been proven above sea level.

An analysis of pyrophyllite from Coxheath mountain, west of Sidney, Cape Breton, is given in Table VII. There are similar occurrences in the Lake Memphremagog district, in Stanstead township, and in Beauce county, all in Quebec; also in the Avalon peninsula of Newfoundland, where large tonnages are said to be available, with scattered lenses of nearly pure pyrophyllite being found (8). More recently reported deposits are in Burin peninsula, Newfoundland, of unknown but probably limited extent and of yet undetermined grade.

Unfortunately, most pyrophyllite rocks are interbanded to a greater or lesser extent with low-grade material, and by the very nature of the mineral and its associates they tend to be highly siliceous. Analyses of some samples are given in Table VII.

Finally, mention may be made of deposits, chiefly of sulphates, formed by a variety of processes. Deposits of these types are rare in Canada and those that are known are of small size; they appear to have little interest as potential sources of aluminum. In addition to the natro-alunite on Kyoquot sound, Vancouver island, already referred to, a few other examples have come to our attention. These include occurrences of pickeringite and halotrichite in slaty rock near The Pas, Manitoba, and of pickeringite in Hants county, Nova Scotia; they are little more than mineralogical curiosities.

TABLE VII.— ANALYSES OF METAMORPHIC ROCKS

	I	II	III	IV	V	VI
L.O.I.	2.7	4.3	5.0	3.2	6.5	5.0
SiO_2	56.3	53.4	67.6	71.9	50.3	65.0
TiO_2	1.0	0.9	0.7	n.d.	n.d.	n.d.
Al_2O_3	24.8	29.8	22.6	23.6	32.6	28.7
Fe_2O_3	7.0	0.8	2.2	0.1	tr.	0.2
MgO	n.d.	0.6	n.d.	n.d.	1.2	0.2
CaO	0.8	tr.	0.3	n.d.	n.d.	nil
Alk.	n.d.	n.d.	n.d.	0.8	n.d.	0.7

I — Andalusite schist, Purcell cove, Nova Scotia.
II — Muscovite schist, Sioux Lookout, Ontario.
III— Pyrophyllite schist, Coxheath mountain, Cape Breton.
IV— Pyrophyllite-quartz rock, Kyoquot sound, Vancouver Island.
V — "Agalmatolite from Stanstead Township, Range 1, Lot 15" (8).
VI— "Best grade of pyrophyllite, from Mine Hill Quarry", Newfoundland (8).

Conclusions

The winning of aluminum for Canadian metal production from silicate and other minerals is at present not much more than an academic problem, although a few years ago it held top priority on laboratory schedules. As long as bauxite can be imported it will undoubtedly be cheaper to treat than any domestic ores. Furthermore, certain high-alumina materials are already used by other industries, and the aluminum industry would have to compete with them if local ores were to be forced into use. The more important factors governing or dictating the use of such raw materials include the following:

(1) World conditions
(2) Bauxite reserves
(3) Metal consumption and requirements
(4) Grade of the possible ore
(5) Accessibility, transportation, power supplies
(6) Availability of other raw materials required for the process

The relation of alumina to other constituents of the ore will be important; alumina can be recovered from some compounds more easily than from others. A rock with medium alumina and high alkalies (such as the nepheline rocks) might be preferable to one with high alumina and low alkalies. On the basis of our present knowledge of aluminum-bearing rocks other than bauxites, the most likely Canadian sources of aluminum in a case of emergency or radically modified economics appear to be anorthosite and nepheline syenite. Both these rocks have been successfully processed on the pilot-plant scale. There are enormous bodies of both rocks in Canada, and anorthosite is one of the most abundant types adjacent to the great Arvida power and production centre.

Clays are treatable by more than one method, and alumina can be recovered from them, but the results of our surveys show that they cannot have the same potential importance for the aluminum industry of Canada as they appear to have for that of other countries; as a consequence, their utilization has been given less emphasis in process development studies.

Acknowledgments

The information in this paper has come from many sources, and it is impossible to acknowledge each one separately. We are especially indebted to personnel of the various provincial Bureaus of Mines or Geological Surveys for assistance and information.

Grateful acknowledgment is made to Aluminium Laboratories, Limited, for permission to publish this paper, and to Dr. E. C. Harder for discussion during its preparation.

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DISCUSSION

ON

Backfilling at Noranda*

By F.E. PATTON

(*Transactions, Volume LV,* 1952, pp. 360)

Mr. J. R. Giegerich (Consolidated Mining and Smelting Company, Ltd.): Is the combination 25 per cent coarse, 72 per cent fine, and 3 per cent pyrrhotite an optimum ratio of mixture?

Mr. Patton: The higher the fines the better, but there is very little segregation in the above mixture.

Professor A. V. Corlett (Queen's University): Is there any expansion or shrinkage in the backfill due to chemical changes?

Mr. Patton: There is a slight shrinkage, but it is made up by oxidation.

Mr. E. P. Thompson (Hollinger Consolidated Gold Mines, Ltd.): How does this type of fill rate as a ground stabilizer?

Mr. Patton: No rock bursts occur at Noranda, and the fill is excellent for ground support.

Mr. A. H. Miller: What is the ratio of backfill to ore?

Mr. Patton: The ratio is two to one by volume.

*April 1952 Bulletin, pp. 191-197; *Transactions,* Volume LV, 1952, pp. 137-143.

Geology of the Coleville Field, Saskatchewan

*By G. J. HAMILTON**

(*Petroleum and Natural Gas Division, Edmonton, May,* 1952)

(*Transactions, Volume LV,* 1952, pp. 361-365)

Introduction

THE Royalite-Canada Southern Coleville No. 1 gas strike of August 15th, 1951, and the closely following low gravity black-oil strike of August 23rd, were the first of a series of recent oil and gas discoveries in western Saskatchewan. They are in what is now known as the Coleville field, 25 miles north and west of Kindersley and 30 miles east of the Alberta-Saskatchewan border (Figure 1).

Gas flows have been recorded in the Viking sand at daily rates of between 5 and 10 million cubic feet, and black oil of 15° A.P.I. gravity is produced from Mississippian sediments of Banff age at a daily rate of 50 to 100 barrels on pump.

Development drilling was carried on during the winter months, with very little attempt at step-out exploratory ventures. This year, however, a step-out programme to the northeast (Figure 2) is being carried on in order to determine the position of a possible truncation of Mississippian sediments in that direction. Section 30 of the Coleville area has been fully developed on a 20-acre wellsite basis, and structural information is restricted to local rather than regional interpretation. No attempt will be made in this paper to interpret the Coleville area with respect to the regional geology in any but a general manner.

* Royalite Oil Company, Limited, Calgary, Alta.

Geology

Devonian

Discussion of the geological history of the sediments underlying the oil pool need not include strata that antedate the main salt bed (Figure 3). With the subsidence of this regional salt basin, Devonian carbonate rocks were deposited in thicknesses up to 1,800 feet. Fluctuations in the level of the sea floor occurred as evinced by anhydrite beds and in some areas by minor salt beds. No evidence has been found, however, to indicate that complete exposure of the sediments ever took place. The first major uplift after the Elm Point age occurred with the advent of Potlatch time.

The Potlatch is underlain by beds of probable Jefferson age consisting of porous oil-stained dolomites. These are possibly eroded to some extent, and local structural features in overlying sediments are probably reflections of this subsurface. Post-Mississippian uplifting may also have created structural relief.

The Potlatch member consists of beds of anhydrite, rust-red, oxidized shales, and minor dolomites, suggesting variations in the sea floor of the basin, with evaporite conditions being predominant. The Potlatch member is overlain by the Three Forks shale formation and, while deposition was continuous, the contact of the two beds shows minor re-working, as might be expected.

The Three Forks member represents shallow-water marine conditions, with resulting deposition of green, muddy, fossiliferous shales. The section varies from muddy, slightly fossiliferous shales to indurated and very limy shales containing abundant fossils which in places are in the form of massed shell beds. These variations in sedimentation are probably due to seasonal changes of climate as well as

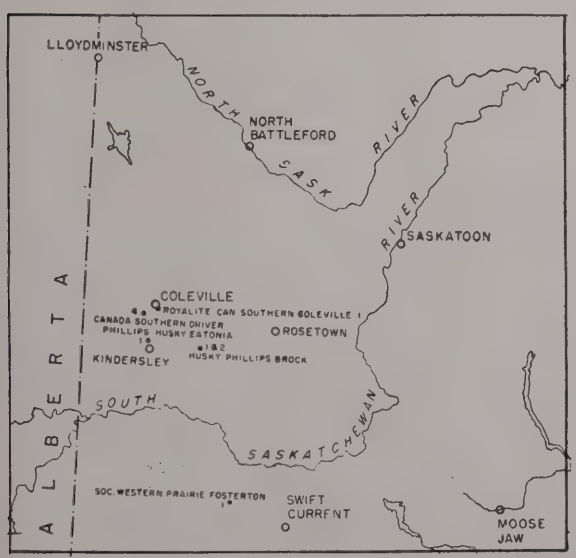
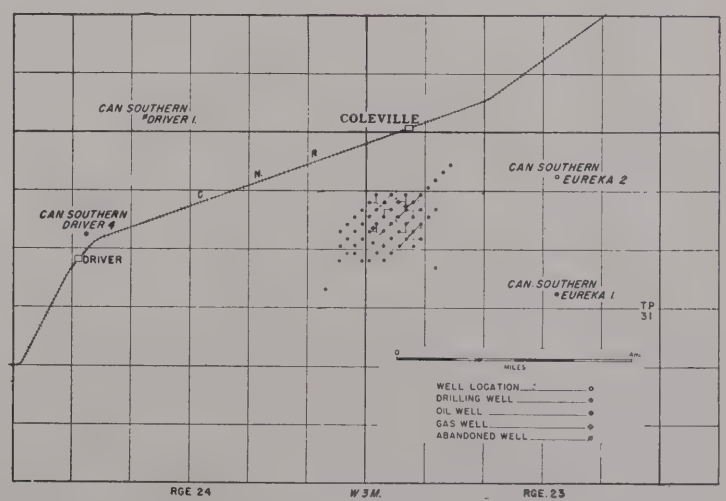


Figure 1.—Sketch-map showing location of Coleville field.



COLEVILLE DISTRICT

Figure 2.

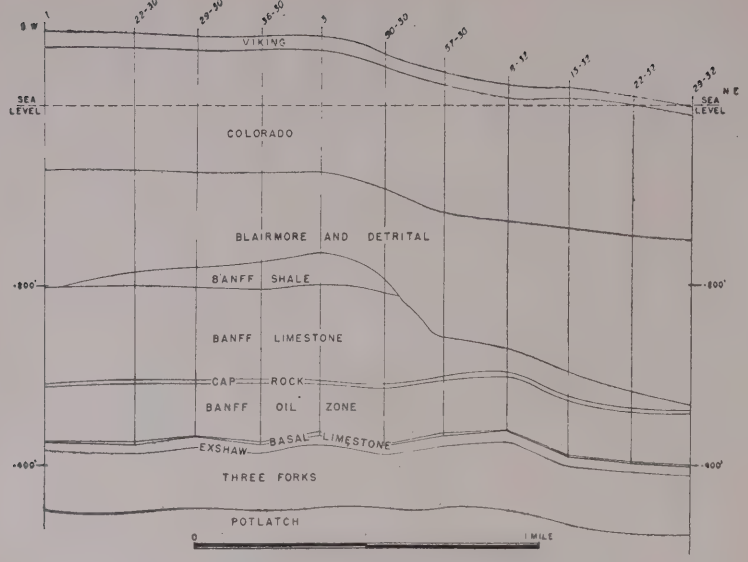


Figure 3.—Structure cross-section, Coleville field.

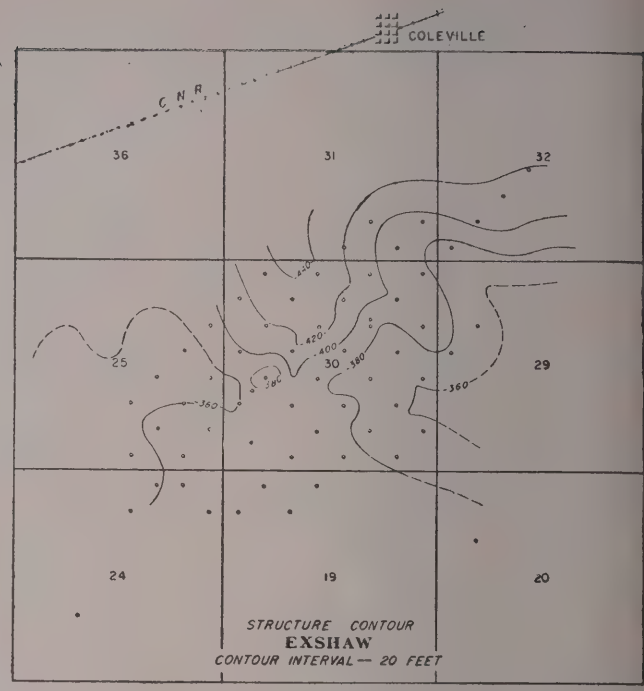


Figure 4.

to slight vertical movements in the depositional surface. The overlying black-brown Exshaw shales appear conformable with the Three Forks shales and rest on this latter member with no trace of unconformity.

The wide areal extent of the Exshaw shale is indicative of a stable shallow-water condition. These shales are black or brown, fissile, and contain numerous plant spores which are characteristics of the member elsewhere. The section includes tough, indurated, fossiliferous, limy zones, and is remarkably similar to the Three Forks shale, with the exception of colour.

Mississippian

Overlying the Exshaw shale is a thin limestone bed, two to ten feet in thickness. This bed is formed of grey-green, cherty, oolitic, and highly fossiliferous limestone. That it may have been greater in thickness at one time is indicated by the reworked condition of the upper contact, where the limestone is cross-bedded and intermixed with the overlying Banff oil sand. The varying thickness indicates that the bed may have been partially removed by the scouring action.

The Banff oil zone averages 60 feet in thickness over the field and consists of sand, sandstone, and shale. The sandstone is generally present in the lower half of the section and consists of well sorted quartz grains, some glauconite, and pyrite. This section is interbedded and cross-bedded with green shale stringers, and some thicker bands of black oil shales are present where sand development is poor. The sandstone has many horizontal partings and, while the whole is oil-stained, the maximum oil production of this zone is probably from these partings. Where encountered in some of the wells, the base of the sandstone is highly indurated.

The upper oil zone consists of sand and green shale bands. The sand is unconsolidated and made up of well sorted quartz grains. This zone is saturated with black oil. Throughout the oil zone there are fragments of limestone containing fossils and carbonized plant remains. These are possibly fragments that were plucked from the basal lime section, and the carbonaceous material they contain is evidence of trapped shallow-water marine vegetation such as seaweed.

In wells drilled outside the Coleville field it is very evident, except over high areas of the sea bottom, that the Banff oil zone tends to become shaly or calcareous. A local high in the region of the field can be assumed over which the winnowing of sediments created a condition favourable to sand deposition along with minor stringers of argillaceous material.

A shale and limestone sequence immediately above the oil zone constitutes a competent cap-rock. The limestone is buff or brownish, dense, crystalline, and generally non-fossiliferous, and the shales are varicoloured, green, brown, and black. Three distinct shades of green are represented, while the brown shale band is strikingly similar to the Exshaw shale. Much re-working has occurred, with cross-bedding and inclusion of one type of sediment in the other.

Next in geological sequence are the Upper Banff shales and limestones. The formation, while named Banff here, may be a clastic equivalent of carbonate Madison sediments to the southeast. The black, dense calcareous shales of the member are interbedded with dense, argillaceous, crystalline and fossiliferous limestones; the limestones being predominant at the top and the shales at the bottom of the 120-foot section. The fossils are chiefly crinoidal. Overlying the Banff in the central portion of the field is a vari-coloured shale member which, while likely of detrital origin, could be Banff because of its position and regular stratigraphic occurrence.

The Mississippian sediments were possibly not much thicker than they appear at the present time and, although truncation of the formation has occurred to the near north, it is assumed that here the present line of truncation is very nearly the extent of original Mississippian sedimentation. The uplifting movement was stronger to the north, although some general uplift occurred to the south. The stronger northern uplift created a monoclinal crest at, or north of, the Coleville area, thus creating a condition favourable to truncation.

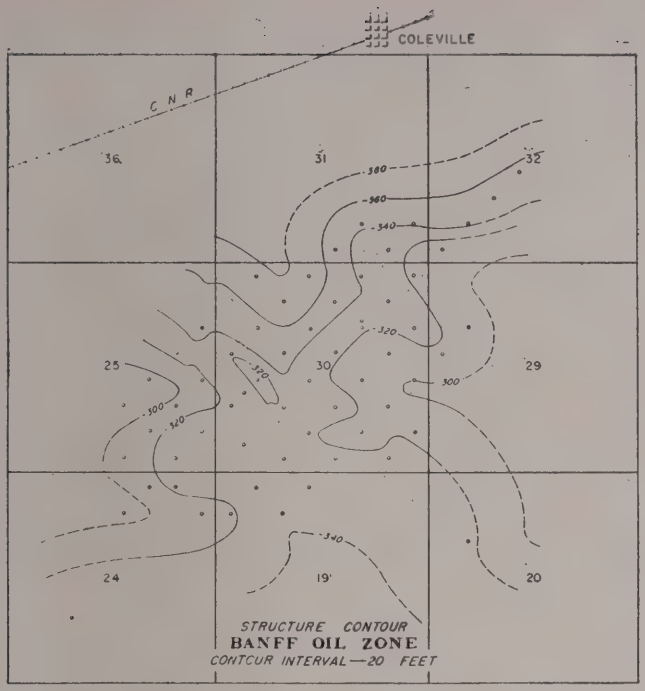


Figure 5.

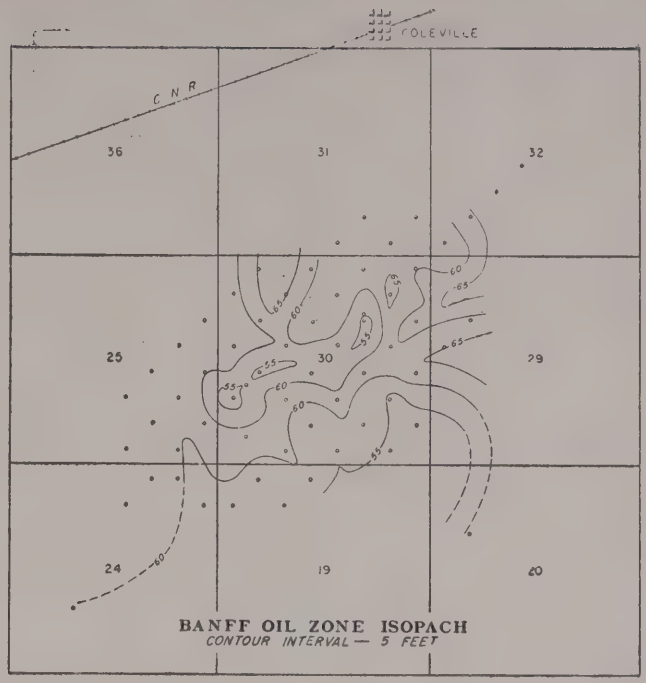


Figure 6.

Cretaceous

Post-Mississippian detrital beds blanket the erosion surface, with typical thinning over the highs and thickening in the lows. These sediments are a heterogeneous mixture of oolitic chert and dolomite, sand, varicoloured shales, chalky limestones, and cherts. The contact between detrital and Blairmore beds is quite distinct in the Coleville area, but becomes obscure as basin conditions are approached.

The Blairmore is predominantly shale with minor shaly and silty sandstone beds over the higher portions of the Mississippian remnants. Thickening, and increased sandstone content, occur in the northern basin areas. Black, dense, bentonitic Inoceramus-bearing Colorado shales of Upper Cretaceous age, 150 feet thick, overlie the Blairmore. These shales are possible source beds for the petroleum and natural gas found in the Viking sand.

The Viking sand member is transitional above and below with the Colorado shales. Where regional and/or local highs occur, sand development, by winnowing processes, is best. The sand attains a maximum thickness of 19 feet. It is fine grained, quartzose, glauconitic, and pyritic, and has a strong petroleum odour and some oil staining. Where it was deposited in deeper waters, it is shaly.

Black, bentonitic shales of Colorado and Lea Park age, 1,600 feet thick, overlie the Viking sand. The uppermost 600 feet of sediments are sandy shales of the Bearpaw formation, overlain by glacial drift.

Structural Geology

The structure of the Mississippian sediments underlying the Coleville field is a direct reflection of the undulating Exshaw subsurface top, which is the control horizon in this area. The Post-Mississippian strata also reflect these primary structural features but their rôle has been either moderated or accentuated by Post-Palaeozoic erosion and sedimentation. This erosion may have accentuated topographic highs through further scouring of lows by submarine currents and tides.

The principal feature noted on the Exshaw contour surface (Figure 4) is an east-west crest, higher to the east and parted in the centre by a re-entrant on the northern slope of the crest. The southern slope of the crest is relatively flat-lying and featureless, but the northern slope is sharper and drops off some 50 feet in this limited area. This reversal is evident along the mile and a half long northern side of the field. The re-entrant mentioned extends southward through the west-central side of the field, and the overall picture shows a high area that has been channelled by a north-south current.

The contours of the Banff oil zone upper contact reflect the Exshaw contours with remarkable similarity (Figure 5). The 60-foot sand section varies only 10 feet in thickness throughout the field and the parallelism of bedding with the Exshaw and overlying Banff limestone and shale suggests that the present structure was nearly absent at time of deposition. Small minor variations between these two subsurfaces are noted, with the upper sediments having a slightly more accentuated relief. This would be expected of the more coarsely clastic sand horizon.

The thickness of the oil zone (Figure 6) follows the contour pattern closely and therefore some original measure of structure was present before the Post-Mississippian uplift. The section thins along the crest, becomes thicker immediately south of the ridge, and thins again farther south. The zone is generally greater in thickness to the north, with thinning and thickening occurring as expected where outwash conditions are effective.

Structure contours on the top of the Banff Limestone (Figure 7) follow the pattern of those on underlying horizons, but greater relief is present due to accentuation of the earlier topography, as previously mentioned. The central portion of the east-west uplift remains as a series of isolated hills. The northern reversal remains constant where the maximum Banff section is present, but becomes sharper where truncation occurs.

The maximum thickness of post-oil sand sediments is 130 feet, with dif-

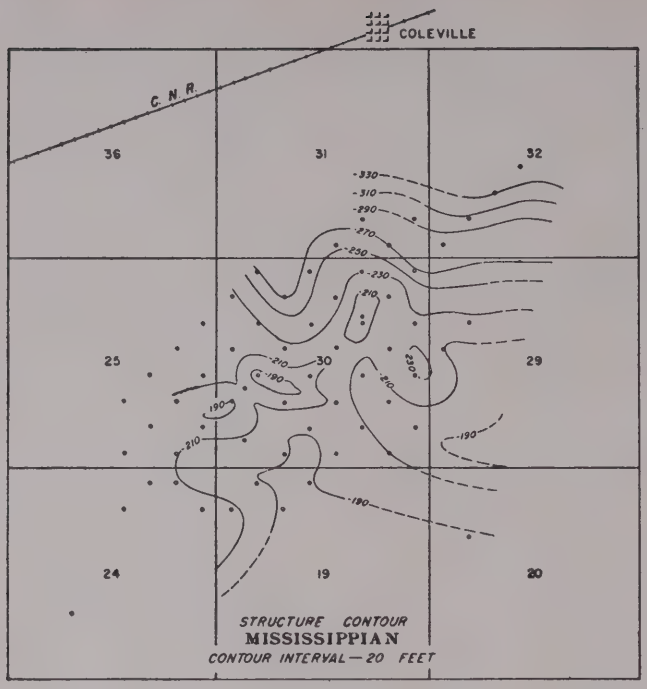


Figure 7.

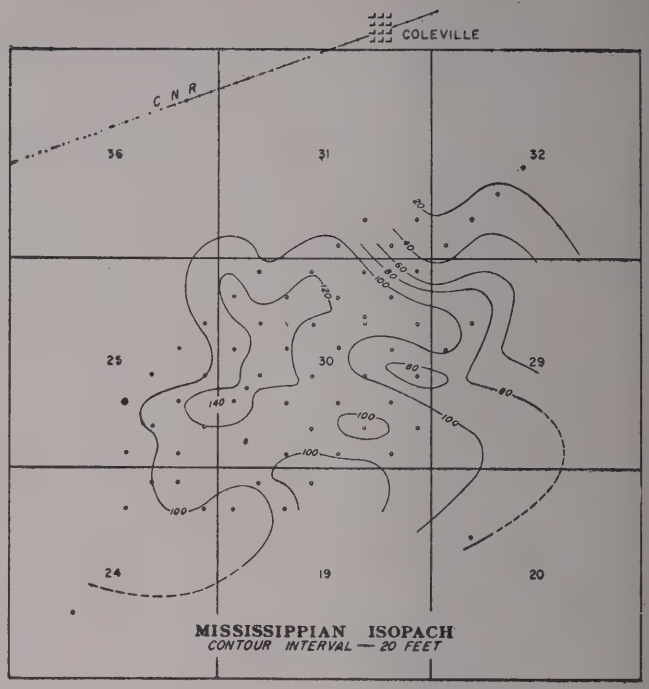


Figure 8.

ferential erosion causing thinning to the north and south in the pre-established lows (Figure 8). Truncation has thinned the beds to 10 feet in the northeast direction, and there has been some truncation, generally on a lesser scale, to the east and south. A sharp cut-back occurs on the east side of the field and 50 feet of sediments have been carved away in this area.

The Blairmore structural top (Figure 9) is similar to the other horizons but, as would be expected, the pattern is moderated because the eroded Banff top was masked by detrital beds that tended to fill-in the depressed areas. The central crest and accompanying north and south slopes are readily evident. The overall thickness (Figure 10) of the Lower Cretaceous and detrital sediments is greatest in the low basin areas, particularly to the northeast where complete truncation must be assumed and large basin conditions are approached.

Structure on the Viking top (Figure 11) lacks the relief noted on the lower horizons; this is to be expected, as the underlying marine Colorado shales mask the Blairmore structures. The features noted below are present, however, but with less relief and sharpness. Decrease in elevation to the north and northeast continues quite sharply. The sand thickness (Figure 12) varies but slightly, with the exception of rapid thinning to the north and northeast where shaling of the sand occurs. A southwest-northeast trend is noted

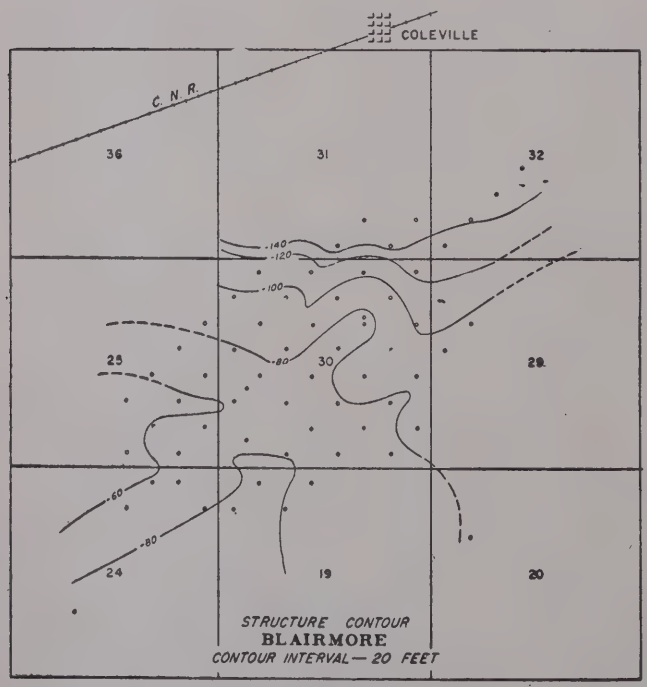


Figure 9.

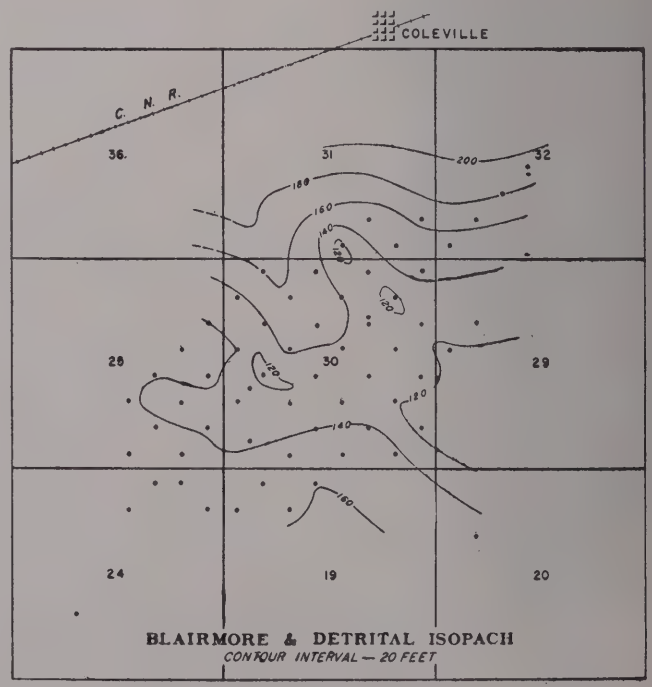


Figure 10.

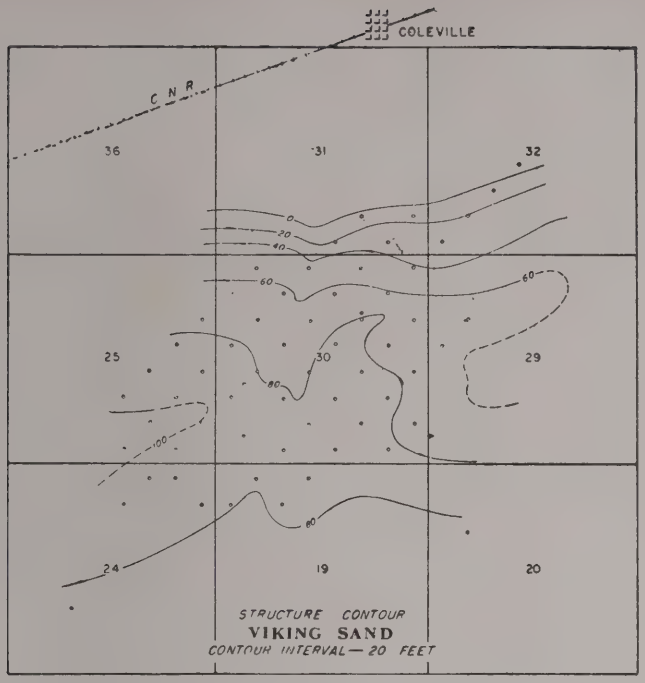


Figure 11.

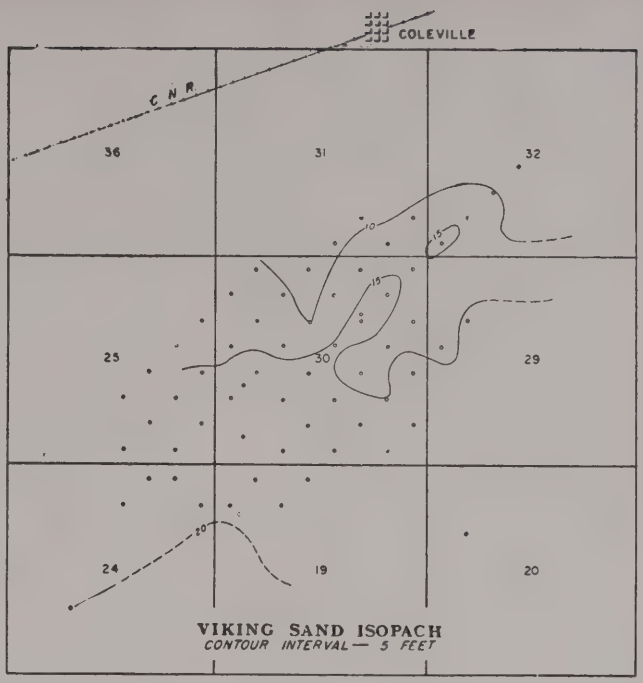


Figure 12.

in which slightly thicker sands occur. The best sand development is on the gentle southern slope, where the sands were deposited from waters moving northeast over the edge of the escarpment.

Conclusions

The Coleville two-zone oil and natural gas discovery indicates that sedimentation in this field was controlled to a remarkable extent by structure and topography. Whether these features are local or widespread is at present unknown, due to insufficient field data. Much further exploration will be required before any estimate can be made of the oil and gas reserves of the field.

DISCUSSION

ON

Operation of the Creighton Mill*

By THE MILL STAFF

(*Transactions, Volume LV,* 1952, pp. 366)

Mr. C. G. McLachlan (Noranda Mines, Ltd.): What is the tonnage of concentrates per day?

Mr. J. E. Lee: 2,000 tons.

Mr. H. R. Banks (Consolidated Mining & Smelting Company, Chapman Camp): What is the number of employees in the mill?

Mr. Lee: One man per 100 tons production; per shift, 20 men: crushing, 4 men; grinding, flotation, and pumps, 2 men each; pumping stations, 9 men.

Mr. J. F. Myers (Tennessee Copper Company): What is the length of time between feeder and weighing machine?

Mr. Lee: Conveys 200 feet at 150 feet per minute — 1¼ minutes.

Mr. Higgins: What is the power on the Symons and short-head crushers?

Mr. Lee: Symons, 200 to 250 h.p.; short-head, 300 h.p.

Mr. Myers: Are the openings of the mills determined experimentally or by catalogue?"

Mr. Lee: Openings mentioned in the paper are the actual openings.

Mr. C. S. Gibson (Mines Accident Prevention Association, Timmins): Does heat come from the mill proper?

Mr. Lee: Heat comes from steam heaters.

Mr. Higgins: What is the size of rods and the clearance?

Mr. Lee: Rods are 3-inch diameter and there is 6-inch clearance to liners.

*May 1952 Bulletin, pp. 253-259; *Transactions,* Volume LV, 1952, pp. 163-169.

DISCUSSION

ON

Mining at Quemont*

By A. G. BALLACHEY, W. D. JAMIESON. and J. R. HAMBLETON

(*Transactions, Volume LV,* 1952, *p.* 366)

Mr. C. S. Gibson (Mines Accident Prevention Association of Ontario, Timmins): To what extent are rock bolts recovered and used again?

Mr. A. G. Ballachey: Very few are recovered in useable condition. They are usually badly twisted. The twisted rods may be used, in part, to make up bolts, etc., or else they reach the scrap metal pile. Nuts and bearer plates are usually recovered. The possibility of straightening the bolts and using them again has recently been under discussion. It should be thoroughly investigated if a tool can be devised for straightening.

Mr. John Kostuik (Barnat Mines, Ltd., Malartic, Que.): We have experienced a lot of broken flooring in cut-and-fill stopes below and ahead of the breasts when the blasts are detonated by millisecond delay caps. Does Quemont have any such trouble?"

Mr. W. D. Jamieson: We do not find broken flooring a serious factor. The toe of the ore pile extends well under the breast to be blasted, so that it, rather than the floor, takes the shock.

A Member: Please indicate the extent of increases in air temperature due to the setting of the fill.

Mr. Ballachey: The fill heats during the oxidation period of the pyrrhotite. Temperature rises are very variable. We lay pipes in the fill at various levels and through these temperatures are recorded. The highest recorded to date was about 160 degrees F. in a large sub-level stope that was filled rapidly after mining was completed. After oxidation is completed, the temperatures gradually drop again. So far, this heating has not affected the general temperature of the mine air, because the ventilation is adequate, and the heat has usually been released into the exhaust air currents and passed rapidly from the mine

Local hot spots in fill have occurred in cut-and-fill stopes, which temporarily cause the air in these stopes to be somewhat warmer than the average. As these stopes are filled intermittently with moderate quantities there has been little build-up of heat.

Mr. Cyril T. Young, Toronto: Is corrosion an important factor in the use of rock bolts?"

Mr. Ballachey: No. The time period is not long enough for corrosion to be a problem.

*January 1952 Bulletin, pp. 58-69; *Transactions,* Volume LV, 1952, pp. 57-68.

Mechanical-Metallurgical Problems Associated with Mine Drilling Operations*

By T. W. WLODEK†

(*Transactions, Volume LV*, 1952, pp. 367-373)

ABSTRACT

Satisfactory mine drilling operations depend on the good performance of the drilling machine, and of what may be called the drill set, *i.e.*, the following four elements: bit, attachment, drill rod, and shank end. The field performances of these four elements in the drill set are analysed and their service lives are compared. The most common causes of service failure are reviewed and possibilities of increasing service life are discussed.

The laboratory methods now employed for the simulated service testing of tungsten carbide bits, conical push-on and threaded types of attachments, drill rods, and shank ends are described, and some results are discussed. The objectives of the research programme now being carried out on the above mentioned elements are outlined.

A number of mines in British Columbia, northern Ontario, and northern Quebec have been visited, and, with the co-operation of the C.I.M. Branches in these areas, meetings with representatives of the mining and steel industries concerned have been held. In this paper, the mechanical-metallurgical problems most frequently encountered and discussed during these visits and meetings are described and analysed.

* * * *

Introduction

SUCCESSFUL mine drill operations depend to a great extent on the satisfactory field performance of the four elements which form the mining drill set, shown diagrammatically in Figure 1.

These important elements are:

1. — *The bit,* which may be one of the following types:
   - (*a*) Detachable tungsten carbide bit
   - (*b*) Inserted rod bit, so called chisel bit
   - (*c*) Detachable steel bit
   - (*d*) Solid rod bit

2.—*The bit-end attachment,* of which the three most prominent types are:
   - (*a*) Conical or push-on attachment
   - (*b*) Threaded attachment
   - (*c*) Inserted solid rod attachment

3.—*The drill rod,* of various shapes and dimensions.

4.—*The shank-end,* collared or simple type.

The particular types of each of these four elements which are selected to form a drill set vary from mine to mine. The selections are made with the aim of obtaining the most economical performance under the particular drilling conditions encountered and with the equipment available.

The manner of failure and relative life of each element in the drill set change with the drilling conditions encountered and with the equipment available, and therefore the design of these elements and the selection of the proper drill set for each drilling condition become difficult problems.

In seeking the solutions of these problems, there are two methods which may be employed. Data on the field performance of each element may be recorded and causes of failure determined. Another method of evaluating the various elements in the drill set would be to test them under simulated working conditions in the laboratory.

A clear overall picture of the performance of the drill set, *i.e.*, bit, attachment, drill rod, and shank end, could be obtained from the recorded field data as in Table I.

These data set up, as given for the example in Table I, for the various mining and operating conditions, together with the mechanical-metallurgical data as specified below, should give quite a complete picture of the technical performance of the drill set. In order to agree on a certain uniform method of recording efficiently and properly the field performance of drill steel, it is necessary to consider and record the following data:

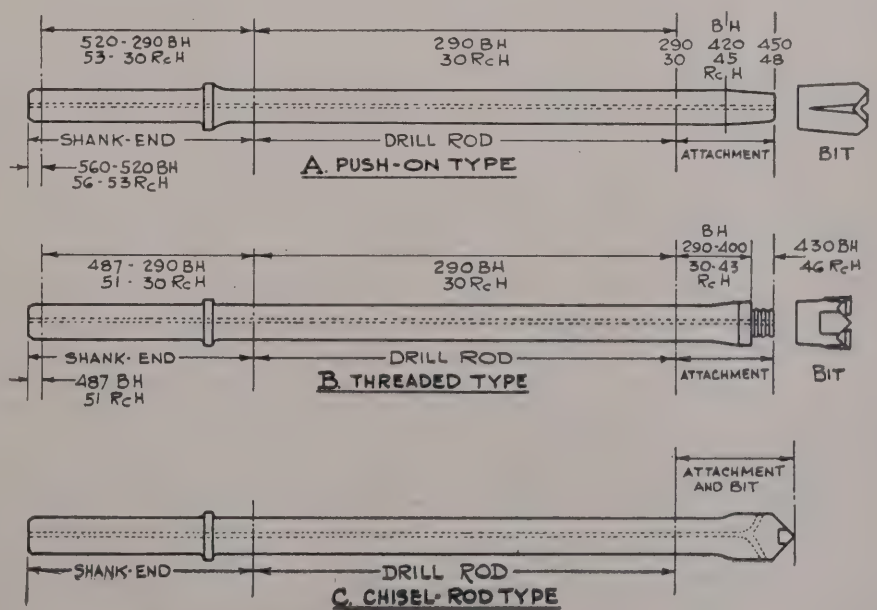


Figure 1.—Schematic diagram of drill set. Alloyed steel.

*Published by permission of the Director-General of Scientific Services, Department of Mines and Technical Surveys, Ottawa.

†Research Metallurgist, Division of Physical Metallurgy, Mines Branch, Ottawa.

A.—Mechanical-Metallurgical

1.—Data on forging and up-setting operations of both ends of drill rod, *i.e.*, bit end and shank end.

2.—Complete data on heat treatment of bit end and shank end.

3.—Data on surface finish of the attachment, drill rod, and shank end.

4.—Hardness gradient and hardness range of drill rod at attachment and shank end.

5.—Allowed tolerances of, and exact contours of thread of, the attachment, fillets of the attachment, and shank end.

B.—Mining Conditions

1.—Data on hardness and type of rock and ore drilled, *e.g.*, expressed in the length of hole drilled and speed of penetration, using the same diameter and type of a standard throw-away steel bit and 3½-inch or 3-inch diameter drilling machine.

2.—Corrosive properties of the mine water used.

3.—Type of drilling operations, drifting, cross-cutting, stoping, or sinking.

C.—Operating Conditions

1.—Piston diameter of the drilling machine used, type, weight, number of blows per minute, and energy of a single blow in foot-pounds.

2.—Approximate air pressure.

Calculated percentage ratios of bit, attachment, shank end, and drill rod life are the most significant figures, showing where the weakest point in the drill set is located.

Figures given in Table I on bit life, attachment life, drill rod life, and shank-end life should be supplemented by detailed information on the type of corresponding failure in a manner similar to that used in this paper. This information should also give the percentage of the several established types of failure encountered. In the field performance of all four elements of the drill set, the following mechanical-metallurgical effects could have an important influence:

1.—*Surface imperfections* of geometrical and metallurgical origin introducing local stress concentrations:

A—Irregularities of the surface contour, sharp deep notches of as-rolled, heat-treated, and forged surfaces.

TABLE I.—FIELD PERFORMANCE OF DRILL SET

MINING AND OPERATING CONDITIONS		DRILL STEEL	DRILL BIT	BIT	ATTACHMENT	ROD	SHANK END			REMARKS
1 CHARACTERISTICS OF ROCK & ORE	1 AIR PRESSURE Nº OF BLOWS	1 DIMENSIONS		1 TOTAL LIFE (FEET)				1 PENETRATION INCH / P.M	1 Nº OF REGRINDINGS OF BIT	
2 TYPE OF MINING OPERATION	2 DIAMETER OF CYLINDER	2 TYPE		2 PERCENT OF DRILL ROD LIFE				2 IMPACT RESISTANCE OF ROCK (POUNDS)	2 LENGTH OF RODS IN FEET	
3 TYPE OF DRILL MACHINE	3 ENERGY OF BLOW	3 MANUFACTURER		3 TYPE OF FAILURE						
1	1	1	1	1 38·5	1 103	1 500	1 400	1	1	DATA ON MECHANICAL, METALLURGICAL CONDITIONS OF DRILL SET
2	2	2	2	2 7·7%	2 20·6%	2 100%	2 80%			
3	3	3	3	3 W.C INSERT	3 FRETTING CORROSION	3 CORROSION FATIGUE SURFACE	3 CORROSION FATIGUE WATER HOLE	2	2	A.J.P

B—Metallurgical notches introduced by

(*a*) deep penetrations (diffusion) of oxides, either intergranular or random oxidation on the surface of the grains.

(*b*) sharp non-homogeneity of crystal structures of widely different grain sizes and physical properties, such as ferrite, pearlite, martensite, austenite, and a large variety of the intermediate transformation products.

2.—*Internal macro-stresses*

Presence of locked-in macro-stresses left from forging, heat-treatment, brazing operations, and different types of surface treatment such as shot peening, induction hardening, and carburizing.

3.—*Internal micro-stresses*

Magnitude of micro-stresses introduced by cold working of metal (plastic deformation), such as shot peening or by aging, precipitation hardening, and so on.

4—*Fatigue stress corrosion* under mine-water corrosion.

5.—*Chemical composition and heat treatment of steel*

Kind of steel chosen as to the chemical composition and optimum microstructure obtained through a heat-treatment introducing acceptable pattern of thermal macro-stresses.

6.—*Working stresses*

The level of the working stresses which are governed by the choice of proper shapes, fillets, and sections, the size of the drilling machine used, and other mining and operating conditions.

The influence of surface stress-raisers introduced by the presence of mechanical and metallurgical notches combined with the stress concentrations inherent in the design, *i.e.*, sudden changes in sections, fillets, and threads, often overshadow the effect of static and dynamic properties of the crystal structure as revealed by microscopic examination.

All four elements of our drill set are heat-treated by one or more of the methods listed below:

1.—*Normalizing, i.e.,* cooling in the still air from temperatures above Ac_3 takes place in drill steel rod, especially after the completion of rolling operations, after up-setting and forging operations of attachment end and shank-end of drill steel, or as a result of a specially applied treatment to refine the grain size.

2.—*Annealing, i.e.,* slow controlled cooling, either in the furnace or in an insulating medium (mica) down from Ac_3, Ac_1, or lower temperature range. This heat treatment could be applied after forging and up-setting operations and also after normalizing treatment.

3.—*Hardening,* a heat treatment in which the steel is heated over the Ac_3 austenitizing temperature and cooled at different rates as may be obtained by water, oil, or air. Partial hardening, *i.e.,* partial austenitizing, a heat treatment in which the steel is heated to a temperature be-

tween Ac_3 and Ac_1, is sometimes used on drill rods to obtain a non-homogeneous microstructure, *e.g.*, ferrite, and a wide variety of austenitic transformation products, characterized by low thermal residual stresses. Partial hardening takes place in drill steel in a zone between the unheated and heated sections of the rod.

In both hardening treatments, the austenite transformation could take place either while the steel is cooling down to room temperature or while being held at chosen intermediate temperatures. In these heat treatments, a variety of transformation structures could be formed (martensite, bainite, pearlite, ferrite, and carbide). Adding to this variety of possible heat treatments, a number of carburizing treatments used in drill rod, attachment, and shank-end will produce nearly all possible variations of microstructures ranging from ferrite to austenite and of different grain sizes.

4.—*Tempering*, reheating hardened or normalized steel to a temperature below the transformation range and cooling afterwards.

Our task is to select the technical and economical optimum for (1) the shape of our four elements, (2) surface finish, (3) chemical composition of the steel, (4) heat treatment for the required microstructure, and (5) favourable pattern of the residual macro-stresses.

At present, thanks to the great efforts of the mining and steel industries, the four elements in the drill set have reached a high state of development and therefore the starting basis for further research is well established.

The introduction for such a wide research programme should be a simple statement of the mechanical-metallurgical problems existing in mining drill operation and their analysis, and the programme of further development. Since the mining and steel industries in the present stage of mining technique are interested only in reaching the practical optimum performance, it is important, because of the large number of possible alternatives of known and unknown factors, to develop simple methods of testing all four elements under consideration. These methods should simulate the actual working conditions as closely as possible and give reproducible results. Here we will mention the general outline of testing methods designed to simulate the working conditions of all four elements of our drill set.

First Element — The Bit

For the time being we would like to analyse tungsten-carbide bits only, leaving out steel bits which, because of the large quantity used and the low price, could be tested in actual field operations, the average statistical life being the simplest acceptance test. The tungsten-carbide bits, during actual field service, fail in many ways.

A method now employed to test tungsten-carbide bits is to subject them to dynamic compression (push-push) loading under water-corrosion conditions. The severity of the testing conditions is regulated by the magnitude of the dynamic load range and the static preload and by the area of the tungsten-carbide (W-C) tips. The tungsten-carbide bit must be designed for high static and dynamic strength under mine-water corrosion conditions. High wear, abrasive resistance, and impact strength are essential properties of the W-C tips and an extremely strong brazing bond between W-C tips and steel is of basic importance, because the tips should wear down before they become loose.

The steel body, the W-C tips, and the brazing bond between the steel body and the tips, may be points of failure in the tungsten-carbide bit, depending upon the drilling conditions. The designer and manufacturer of the bit is aiming at long life of the bit, high abrasive resistance of the gauge, *i.e.*, low gauge loss per foot drilled, high abrasive resistance of the W-C tips, and a good drilling speed. Because of the complicated nature and severity of working conditions of the tungsten-carbide bit, a detailed discussion on the design and performance of this element will be left to a special report. In this paper, only those features of design and manufacturing procedure which directly affect the fatigue strength of this element will be discussed.

Because of the non-homogeneous structure of the W-C bit, it being composed of a steel body, W-C tips, and braze, air hardening alloyed steels are used in the manufacture of the steel body. Requirements of high strength and abrasive resistance of the steel body could be met only by alloyed steel.

An important point is a favourable pattern of locked-in macro-stresses in the steel body and in the W-C tips. If favourably distributed, they may render a substantial contribution toward good performance of the steel body and the tip itself. Data already on hand (1) show the presence of high compressive stresses in W-C tips which, combined with compression-impact loads to which the tips are exposed during drilling, might produce a favourable resultant stress.

An Avery pulsator fatigue testing machine was employed to test the tungsten-carbide bits. The applied unit stress on the cutting edges is adjusted by the length of the W-C edges supported in a hard steel plate of the testing attachment and by the regulation of the magnitude of the dynamic load and static preload. By reducing the length of the edges of the W-C tips, the applied unit stress could be correspondingly increased. The application of this method is limited by the capacity of the available fatigue machine, which is up to 40,000 pounds. Figure 2 is a close-up view of the machine used. Typical results obtained, in the form of S-N curves, are shown in Figure 3.

The possibility of duplicating the field conditions with this testing method is slight. The stressing and water-corrosion conditions are simulated to some extent, but it is practically impossible to duplicate the abrasive and shock loads to which the tungsten-carbide tips are exposed, at least in an installation of a reasonable cost.

The results obtained to date have been very useful in that they have instigated a number of valuable suggestions as to details of shape, hardness pattern, and surface treatment. This test is continuing and it is expected that the list of present information will be extended.

Second Element — The Attachment

The dynamic performance of the bit end of the drill rod is one of the weakest points in our drill set (as shown in Table I). The relative percentage life of the attachment varies from as low as 5 per cent to as high as 100 per cent of the life of the main element, *i.e.*, the drill rod itself.

Many thousands of dollars are spent yearly for manufacturing adaptors, mainly threaded, push-on, and chisel types. At present, where the mining industry is switching toward the use of W-C bits, the failure of the attachment is quite likely to cause the bit, which is a valuable tool, to be damaged. A failure of the attachment was not so costly

(1) For references, see end of paper.

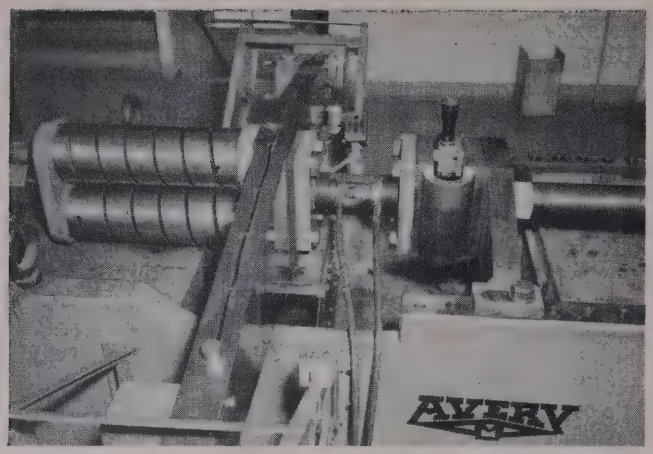


Figure 2.—Avery Pulsator fatigue testing machine, capacity 20 tons.

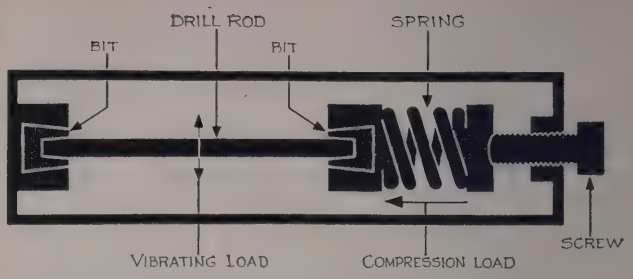


Figure 5.—Adaptor for testing drill rod attachment.

Figure 4.—Typical examples of fatigue failure of conical (left) and threaded (right) attachments.

when the cheap throw-away steel bit was used. Therefore, improvement in the performance of the attachment is a very important matter and a huge effort is being made in that direction.

At present, the life of an attachment varies widely from about 30 to 1,000 feet of drilling, and depends on mining and operating conditions. A typical failure of the push-on conical type of attachment is the failure of the conical part of the attachment, with the nucleus of fatigue located on the circumference of the conical surface, perpendicular to the longitudinal axis of the drill rod. From this type of fatigue failure (shown in Figure 4, left side), and from the direction of the crack, it could be assumed that the axial tension-compression stresses, combined with fretting and stress-corrosion, are mainly responsible for its occurrence.

Failures of this type of attachment above the conical part which is inserted in the bit are similar to typical failures of drill steel and are of a corrosion-fatigue type. The nucleus of fatigue is usually on the outside surface of the drill steel and is generally located in the heat-affected zone if the conical shape of the attachment was heat-treated, which is the usual procedure, or in the forged and heat-treated zone if the cone was formed by forging operation instead of by simple machining. The same type of failure is also observed on the inside of the water hole of the attachment. The direction of the failure cracks usually indicates that the longitudinal tension-compression stress, combined with the effect of stress-corrosion, are responsible for this type of failure.

Examples of typical failures of the threaded type of attachments are:

1.—Failure on the fillet between the shoulder of the rod and the threaded portion (example shown in Figure 4, right side). There are a number of reasons for this type of failure, a few of which are listed below:

(*a*) Excessive notch sensitivity of the material used

(*b*) Too small radius of the fillet or rough machining

(*c*) Eccentric loading, either because the threaded portion of the rod is off-centre with the longitudinal axis of the rod or because the shoulder of the rod is not in the axis of the thread or is not perpendicular to the axis of the rod

(*d*) Unfavourable distribution of the internal macro-stresses in the fillet

(*e*) Bottoming of the threaded portion in the bit.

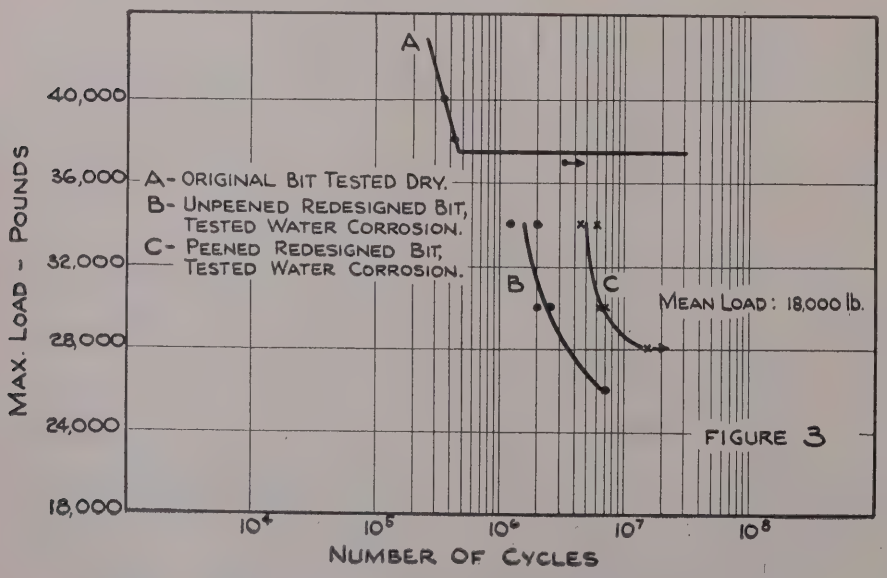


Figure 3.—Fatigue tests on tungsten carbide tipped rock drill bits.

2.—Failure on the shoulder of the attachment, with the nucleus of fatigue failure on the outside or on the inside surface from the water hole. Both types of failure could be caused by the mechanical or metallurgical notch.

For preliminary research on possible mechanical-metallurgical improvements, two special methods have been developed for the simulated testing of attachments. The adaptors used in the first method are shown in Figures 5 and 6.

This method of testing applies the principle of static compression with dynamic bending of the drill rod supported in two experimental

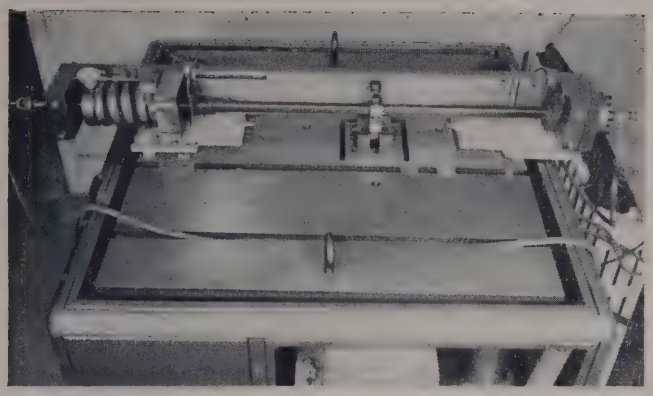


Figure 6.—Baldwin-Sonntag fatigue machine with built-in adaptor shown in Figure 5.

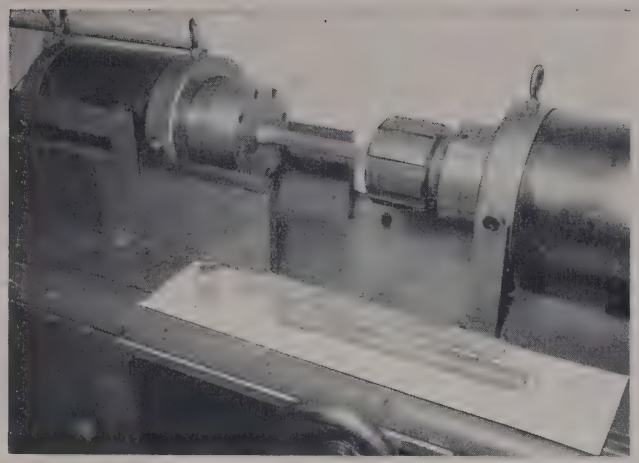


Figure 8.—10,000 inch-pound Sonntag rotating beam machine modified for fatigue testing threaded-type attachments.

bits placed on both ends. The S-N diagram (Figure 7) obtained will enable the life of different attachments under controlled working conditions to be estimated. The absolute and relative values obtained will enable the evaluation of the effect of all the important mechanical-metallurgical factors mentioned before, such as kind of steel, heat-treatment, surface finish, locked-in thermal macrostresses, and type of shims used. This compresion dynamic bending method is intended for use of push-on and threaded types of adaptors.

Fatigue strength of the push-on attachment is substantially influenced by the fretting corrosion which takes place between adjacent surfaces of the bit and attachment. Since many field failures of the attachments are attributed to the effect of fretting corrosion, it is imperative to investigate this phenomenon on a laboratory scale and provide the industry with simple suggestions based on the numerical results.

In order to reduce the fretting corrosion in push-on conical type adaptors, copper shims are used at present. During field investigations in Ontario and northern Quebec mining districts and in British Columbia, two special methods have been developed which may reduce the effect of fretting corrosion and the effect of local stress concentration between the attachment and bit, and their worth will be determined experimentally.

The adaptors used in the second preliminary method developed for research on possible mechanical-metallurgical improvement of attachments are shown in Figures 8 and 9. In this method, the rotating-bending principle is used, and it is proposed only for the threaded type of attachments. From the obtained number of cycles to failure and the

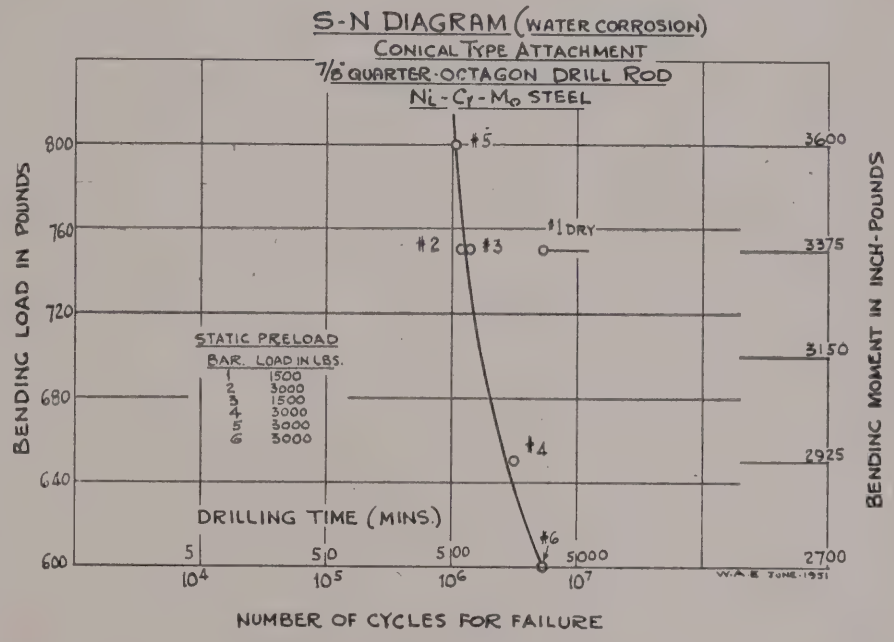


Figure 7.

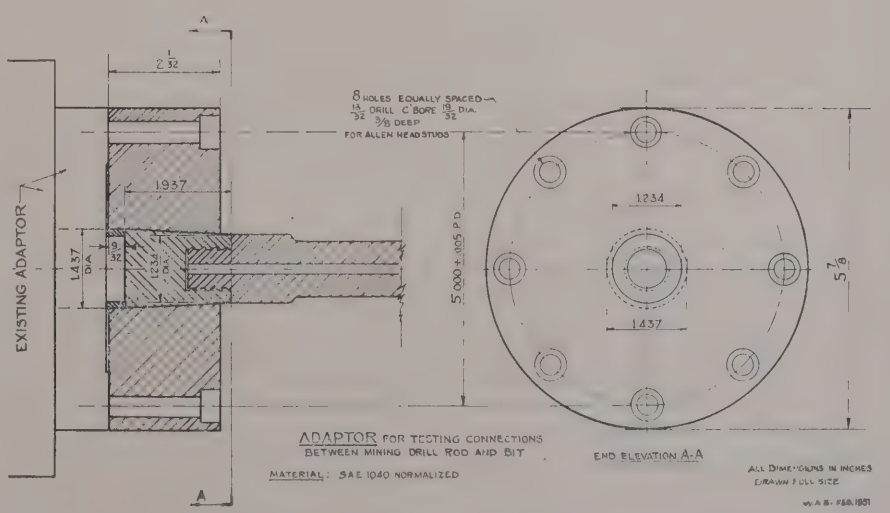


Figure 9.—Adaptor for testing connections between mining drill rod and bit.

bending stress applied, the S-N curves (Figure 10) for different kinds of steels, heat-treatment, surface finish, and shape of thread, and fillet will be derived.

In both proposed methods of testing, the number of cycles of loading could be easily translated to minutes of actual drilling by assuming that, on the average, 2,000 blows per minute are obtained from the drilling machine. The unknown stress level to which the attachment is exposed during drilling operations could be evaluated roughly by comparing the field life and corresponding number of cycles to failure obtained by one of the proposed testing methods. The preliminary test on these methods is being run under dry conditions only, but water-corrosion research is planned.

The load ranges to which these methods will be employed are as follows:

First Method: Compression up to 15,000 lb.
Dynamic bending from 0 to ±4,000 inch-pounds

Second Method: Rotating bending ±10,000 inch-pounds

Third Element — Drill Steel Rods

The preliminary research work on the development of a simple method of fatigue testing full-size drill steel rods has been completed and was discussed during the 1950 C.I.M. Convention in Toronto (2). The fatigue strength of typical kinds of drill steel used in Canada was determined, using the above mentioned method. An account of the preliminary research on the effect of water corrosion and shot peening on the fatigue strength of drill steel and the evaluation of their field performance was also presented at the 1950 C.I.M. Convention (3). The results of research on the effect of induction surface hardening and other surface treatments on fatigue strength and field performance of drill steel were given in another paper, presented at the 1951 C.I.M. Convention (4).

The results obtained are completely reproducible and allow the relative data of the field performance of drill steel to be evaluated.

The endurance strength of drill steel determined by the method presented, when compared with the field performances, will render useful data on our drilling technique.

As an illustration of the application of the data on endurance strength of drill steel obtained by the P.M.R.L. method, one example could be used which was discussed during one of the round-table discussions held at a mine during the field investigations mentioned in this paper.

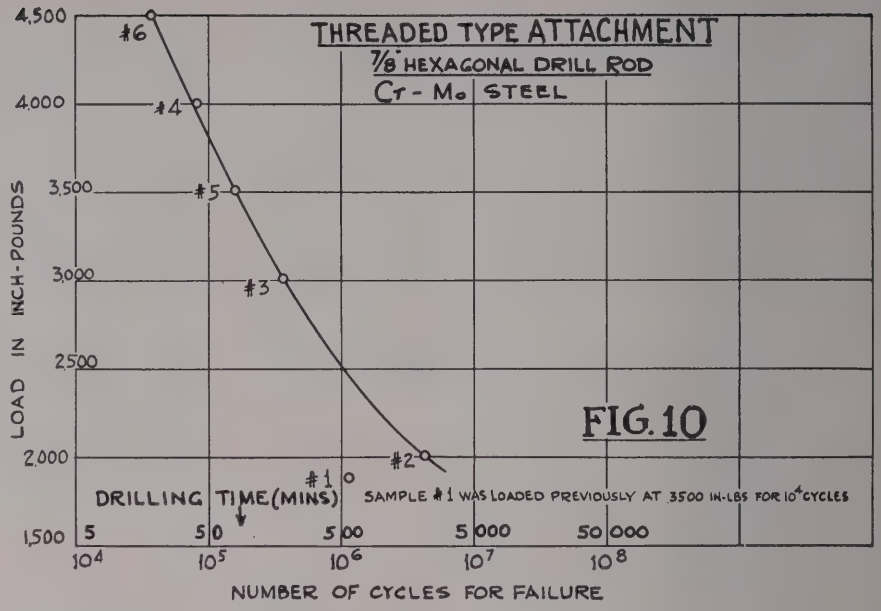


Figure 10.—S-N diagram (dry test), threaded-type attachment.

In a drilling operation in an orebody, a plain carbon steel one-inch quarter-octagon section was used. The drilling machine used was a 3⅛-inch stoper with Craig steel bits, 100 p.s.i. air pressure. Under these mining conditions these drill rods fail, on an average, after 93 feet of drilling.

Field Observation

Drill rods failed:
(1) At attachments .. 8.7 per cent
(2) At the centre of the drill rod ..... 82.9 per cent
(3) At shank end .. 8.4 per cent

Corrected footage per drill steel:
93 ft. × 100/82.9 = 112 ft.

Average drilling speed:
14 in. per min. = 1.16 ft./min.

Time of drilling:
112/1.16 = 96.5 min.

Number of blows of drilling machine per minute = 2,200

Total number of blows (N) = 96.5 × 2,200 = 212,300 cycles

From the S-N relations obtained for one-inch quarter-octagon plain carbon steel *SAE*1080 from the data given in the paper referred to (3), the endurance strength (damaging stress) for 220,000 cycles (Curve *A*) is about ±45,000 p.s.i.

During actual mining operations, the drill rod was exposed to different magnitudes of stress but the effective stress level could be accepted as ±45,000 p.s.i.

Data obtained from the discussion contributed by Mr. F. R. Anderson, Gardner-Denver Company, Denver, Colorado (3), reveal that, for a sharp Timken carbide bit with good alignment in a 3½-inch drifter with 80 p.s.i. pressure, the maximum tensile stress on the drill steel at the bit end is about 58,000 p.s.i.

This stress is higher than evaluated from our S-N relations (3) but, taking into consideration the fact that a larger machine was used and making the corresponding correction, we have

$$45{,}000 \times \frac{(3\frac{1}{2})^2}{(3\frac{1}{8})^2} = 56{,}250 \text{ p.s.i.}$$

These figures are in general agreement, when due allowance is made for variations in air pressure used and other operating factors.

Continuing further on the same example, we are prepared to answer the following question: "What would be the life of a 1-inch quarter-octagon rod made from *C*, *D*, *E*, *F*, *G*, and *H* steels for the same operating conditions?"

The service life for these steels would be as follows:

Steel *C* (SAE1080 shot-peened, dry; rf 3):

$$N_C = 5 \times 10^6 \text{ cycles} \times \frac{1.16}{2{,}200} = \text{about } 2{,}500 \text{ ft.}$$

Steel *D* (SAE1080 shot-peened, water corrosion):

$$N_D = 2.5 \times 10^6 \times \frac{1.16}{2{,}200} = \text{about } 1{,}250 \text{ ft.}$$

Steel *F* (Ni-Cr-Mo steel, as-rolled, water corrosion):

$N_F = 1 \times 10^6 \times \frac{1.16}{2,200}$ = about 500 ft.

Steel *E* (Ni-Cr-Mo steel, as-rolled, dry condition):

N_E = infinity, stress below the fatigue limit

Steel *G* (Ni-Cr-Mo steel, shot-peened dry condition):

N_G = infinity, stress below the fatigue limit

Steel *H* (Ni-Cr-Mo steel, shot-peened water corrosion):

$N_H = 5 \times 10^6$ = about 2,500 ft.

Fourth Element—Collared and Simple Shank-Ends

Field performance of the collared and simple shank ends may be analysed statistically in a way as shown in Table I and supplemented with the percentage figures on the types of failures encountered. The critical analysis of such information is the first and simplest step toward possible improvement of the field performance of this vital drill-set element.

The simple, *i.e.*, collarless, shank-end on its outside face is rounded by a simple machining operation or is forged and machined and, as a rule, heat-treated to a high hardness in order to prevent excessive plastic deformation under high impact loads from the drilling machine. A proper hardness gradient along this part should reduce the wear of the shank-end in the chuck of the drilling machine.

The collar on a collared shank-end is formed by an up-setting and forging operation and, after machining, is heat-treated according to the chemical composition of the steel used and hardness level required. The collar-forming operations and heat treatment change the overall characteristic of the original drill steel, which was in the as-rolled condition. These operations usually increase the surface imperfections of the steel, both of geometrical and metallurgical character, and affect the pattern of the internal macro-stresses. Major visual changes are the introduction of sharp and sometimes very irregular changes in cross-section of the forged-on collar, and irregularities of the internal water hole. Inadequate fillets on both sides of the collar and irregularities of the water hole are often responsible for the premature failure of the shank-end.

The field performance and endurance strength of this element in relation to the basic strength of drill steel could be improved by avoiding the use of collared-type shank ends. Use of the collarless type would be the simplest way of improving the endurance strength. If such simplification could not be adopted because of the mining operational requirements, care should be taken to minimize these effects. From the data available from our previous papers, shot peening would be the surface treatment recommended to improve the endurance strength of the collared section of the drill set.

The types and locations of fatigue failures of this element are very similar to those encountered in the drill rod mentioned earlier. The typical failure for the shank-end is that, more often than not, the nucleus of fatigue is located at the water hole. This could be explained by intensive action of water corrosion on the inside surface as compared to the outside surface, which is evenly exposed to wear action that reduces to some extent the surface imperfections. Also, the magnitude of bending stresses is smaller at the shank-end because of the guiding grips of the drilling machine, and therefore the start of the fatigue failure from the inside is more probable at the shank-end as compared to the middle portion of the rod. The gripped portion of the shank-end is exposed to wear which might reach such proportions that the required dimension tolerances would reject that part from further use.

For the laboratory simulated-service test of the shank-end, a method similar to that for the drill rod test is recommended (1). The purpose of this type of test would be the determination of the S-N fatigue curves for typical types of shank-end.

The endurance strength determined for the stress level expected in field service would enable the influence of the most important mechanical and metallurgical effects on the field performance of that element to be evaluated. This leads in a simple and logical way toward the evaluation of the different methods of manufacturing used at present and to be recommended in the future.

Conclusions

The above simplified review of the mechanical-metallurgical problems met with in the drill set in mine drilling operations gives a fair picture of a large number of technical difficulties encountered in that field.

In order to expedite the research and industrial development work on further improvements of the field performance of all elements of our drill set, the following steps are recommended:

1.—As far as practically possible, to use a uniform method of recording the field performance of the elements in the drill set as recommended in Table I, supplemented by the mechanical-metallurgical, mining, and operating data and detailed information on the types of failure and their percentages of appearance. It is highly recommended that the data given in Table I be collected from records kept individually for each element, marked with an identification number. Statistical data obtained as average values of records kept individually give a much truer picture than data obtained from overall workshop statistics, which are often overshadowed by assumptions and simplifications.

2.—To develop simple laboratory methods of simulated service testing in order to study the effect of numerous mechanical and metallurgical phenomena on the field performance and endurance strength of all four elements of the drill set. The methods proposed for the simulated service testing of bits, attachments, drill rods, and shank-ends should enable research work on the effects of one or a chosen group of phenomena listed above to be continued under stable and controlled conditions of loading and testing. Obtained dynamic load and number of cycles, so-called S-N relations, could be correlated with the field research sponsored by the mining and steel industries.

The results obtained from the laboratory methods presented above, combined with the extensive data already accumulated from the research work completed by the mining and metals industries, should contribute substantially toward further development of the most efficient drill set.

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Development of the Science of Grinding*

By L. E. DJINGHEUZIAN†

(*Annual Western Meeting, Winnipeg, October*, 1952)

(*Transactions, Volume LV*, 1952, pp. 374-382)

SUMMARY

In the following discussion, the work of present-day investigators on grinding fundamentals is submitted as grindng criteria. These criteria comprise:

(1) Power
(2) Grindability
(3) 'Eighty per cent passing *n*-mesh'
(4) Total work input
(5) Bond's work index as a measure of comparative grinding efficiencies
(6) Work index as a function of grindability
(7) Grindability as a function of '80 per cent passing *n*-mesh'

Finally, in his last chapter, the writer offers the concept of thermodynamic criteria in grinding. This concept is not a new one, having already been advanced by several previous investigators. However, the writer is formulating this concept as a mathematical law, hoping that the analysis and criticism of his reasoning by grinding investigators will finally lead to the establishment of grinding as a full-grown member of the family of natural sciences.

* * *

"It must be understood that the most important advances in our knowledge of the comminution of material lie in the exploration and development of the underlying fundamental laws, rather than in the accumulation of more empirical data".

F. C. Bond and W. L. Maxson, A.I.M.E., Trans., Vol. 134, 1939, p. 323.

* * *

Introduction

During the past two decades an enormous amount of empirical data has been accumulating on grinding. These data have been obtained by mill operators on grinding ores with a diversity of characteristics, and generally they have been published. However, whenever an operator, who has started milling on a new ore, has tried to find a case in literature which would exactly fit his particular grinding problem, he has failed. Evidently, something fundamental has been lacking in all these empirical data, and, as a consequence, each new operator has had to undertake his own experimenting, get the best results he can, maybe publish these results, and let it go at that. This, in the writer's opinion, has been due to the fact that even the ablest and most inquisitive millman has been forced to look on grinding as an art, and so, except very rarely, has not been able to find the time to stop and think; if he did so, it was rather in a philosophic way, that grinding is a fundamental process, a process that takes place in river beds, on the sea shore, etc.

Nevertheless, when all the data so far published are correlated and examined as a whole, one is struck with the advances that have been made in grinding. The writer (7) had made such a study in 1949, and, when the correlation was completed, it showed that there was nothing heterogeneous about the accumulated data and that when brought together they were in harmony with each other. The writer's conclusion was that, if grinding criteria were defined and were expressed in such a way as to make integration of any data into basic formulae possible, grinding would be established as a mathematically exact science, and grinding worries of the harassed mill operator would be at an end!

The first attempts to rationalize comminution were made by Rittinger and Kick. However, their theories led to endless disputes which did not accomplish much, except possibly clearing the path for subsequent investigators.

The first advance in the theory of grinding was made by Dr. H. A. White (19) of Witwatersrand in 1915. This was followed by the rather remarkable plant investigations of Dr. E .W. Davis (6) in the United States, who analysed the action in a ball mill mathematically in 1919. Considering the status of grinding at that time, Davis' classical paper is outstanding: it was conceived by a scientific mind, bold and unfettered. There is no doubt that Davis' investigations led to better understanding of the grinding phenomenon, and, whenever mill operators took the trouble of studying his paper, improved designs of grinding plants resulted. Thus, a start was made in developing grinding as a science.

Only meagre data on grinding were published during the 'twenties', but the early 'thirties' saw what the writer would like to call the beginning of the golden era of grinding investigations with cylindrical mills. It must be said at the outset that, though the writer does not necessarily consider the cylindrical mill the penultimate in grinding, nevertheless he is not convinced, as are many other investigators, that the cylindrical mill is a very inefficient machine.

Grinding Criteria

1.—*Power*

In 1934, Coghill and his associates (9) defined power as the most important criterion in grinding. They advanced a 'new philosophy' which was stated as follows:

> "Briefly, this 'new philisophy' consists of using power as an index of the mill operation, and investigating grinding from the standpoint of set and induced variables, with emphasis on the latter".

Coghill and his associates determined empirically that horsepower for cylindrical mills 6 inches long varied as the 2.6 power of the diameter, thus confirming H. P. du-Pont's original formula. Contem-

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(7) For references, see end of paper.

poraneously, Fahrenwald (8), experimenting with laboratory ball mills, determined the exponent to be 2.65.

Using the same percent of critical speed, same balls, same liners, same type of feed and discharge and same mesh analysis of both, and same relative ball load, Coghill's exponent 2.6 was confirmed both for horsepower and capacity of commercial cylindrical mills by the Lake Shore staff (14), who further say that "perhaps" both the ball wear and the liner wear vary as the diameter to the 2.6 power.

Recently, when designing tube mills for fine grinding with pieces of ore as pebbles, the diameter to the 2.6 power formula was used at Lake Shore in their calculations. After the mills were installed, the grinding results checked Lake Shore calculations within the requirements of research.

The writer believes that the Lake Shore investigations established Coghill's exponential formula as a law, and, by doing so, created for the first time a basis for consistent work with cylindrical mills.

2.—*Grindability*

Contemporaneously with Coghill, F. C. Bond and his associates (11, 4, 5, 1, 2) have started a long-range series of tests on grindabilities of various ores. For Bond's definition of grindability, the reader is referred to (5), pp. 362-363.

From Bond's definition, the grindability of an ore is evidently a function of several variables, namely: hardness, toughness, texture, degree of shatterability induced by the violent severance of the rock from its orebody, mineral composition, and structure of the minerals themselves. Again, temperature of deposition of the mineral and the rate of its cooling after deposition would affect its grindability. It can also be postulated that the more perfect the mosaic crystal structure of a mineral, the harder it is to grind.

The writer assumes that, so far, it has been impossible to resolve grindability into its variable components. Hence, at present, one has to depend on grindability as determined empirically by Bond's tests and which can be defined as the integral of the above variables. Once this definition of grindability as the integral of its variable components is accepted, it becomes a scientific term.

How grindability may be used as a criterion to foretell power-consumption values for cylindrical mills will be discussed later in conjunction with Bond's (3) new theory of comminution.

3.—*Eighty Per Cent Passing n-Mesh*

The criterion of '80 per cent passing' was first introduced by Prof. A. F. Taggart (15) in crushing.

In stating his empirical capacity formula for crushers, Taggart uses R_{80}, which he defines as "the quotient of theoretical square-mesh aperture that would pass 80 per cent of the feed, divided by the aperture that would pass 80 per cent of the product. It is best determined from screen tests".

Taggart says further that:

> "Choice of the 80-per cent point is based on the fact that the effect of stray coarse material on the curvature of the sizing-test graph has disappeared at that point, while the remaining segments of the sizing curves still have a substantially maximum spread".

In his *The Third Theory of Comminution,* Bond (3) uses the criterion of '80 per cent passing' also for grinding.

To find an empirical confirmation of the validity of this criterion in grinding, a review of the literature has been made. The evidence of the significance of the '80 per cent passing' point appears in *Fine Grinding Investigations at Lake Shore* (14).

Study of the Lake Shore paper shows that the measure of grind in their investigations was based on percent plus-28 microns. Since percent plus-28 microns has varied between 15 and 30—generally it has been near 20—the writer ventures to say that the measure of Lake Shore grind has been actually based on around 80 per cent minus-28 microns, which, purely for convenience, was expressed by the authors in plus terms.

It is pertinent to quote the following statements, based on the results of most exhaustive experimental work, from the Lake Shore paper (14):

> "The point grind—in our case to the percentage +28 — possibly gives a complete picture of the grind without the necessity of surface calculations".

Again:

> "It is a fact that the assay of the tailing corresponds in variation with the +28 m. measurement on any consistent, proper chemical set of conditions of treatment (see Figure 38, page 402). Therefore, we have taken the +28 m. point in the infra-sizer for use as a measure of the grind done, as it appears superior to any other measure investigated, and is infinitely more simple in use".

To the writer, the fact that Lake Shore investigators had decided on the 28 micron factor does not appear to be a coincidence; there is nothing haphazard or arbitrary about it; this empirical factor emerged as a result of their most exhaustive investigations.

Taggart (16) has shown that '80 per cent passing' factor is a criterion in crushing; the writer agrees with Bond, who derives the proof from the log-log plots, that '80 per cent passing' is a criterion also in grinding. As an additional proof of this, the writer submits the empirical data from Lake Shore investigations.

4.—*Total Work Input*

In their paper on grinding investigations at Hollinger, the Mill Staff (11) make a remark worthy of the particular attention of every grinding engineer:

> "Anything that can be done underground . . . that will provide a daily ore supply that is uniform in its size and hardness is a direct grinding asset".

E. J. Pryor, in his contribution to the discussion on a paper by T. K. Prentice (12), stated that:

> "Variation in mode of severance of the rock from its lode or of pre-stressing the working places" affects the response of ore to crushing.

Recently, Fred C. Bond (3) presented a paper, *The Third Theory of Comminution,* at the annual meeting of the A.I.M.E. in New York. The writer is particularly indebted to Mr. Bond for his permission to publish excerpts and summaries from his paper before even its publication.

In his paper, in re-examining the assumptions underlying Kick's and Rittinger's theories, Bond makes the following far-reaching statement:

> "The first of these assumptions to be questioned is concerned with the neglect of the work previously done on the feed particles. Any subdivided material represents work input, and the *total work input* to a material should be the primary consideration, not the

work input increment below a certain feed size". (*Italics are writer's*).

Accordingly, Bond defines the total work input as follows:

"Total work input (W_t) represents the total work or energy input in KWH/ton applied to obtain a size reduction from a feed of theoretically infinite particle size to a definite product size. The total work input is the sum of the work input in reduction of the feed to the product and all work previously put into the feed".

In the opinion of the writer, the new concept of the *total work input* is of basic importance. In enunciating it, Bond is formally stating that comminution starts with the severance of the rock when the miner first inserts his drill into it and follows his drilling by blasting. The severance of the rock represents useful work input, which is a function of the mode of severance. Thus, the first variable which affects crushing and grinding is the mode of severance of the ore, expressed in energy units, and the inclusion of this variable in the new concept of total work input is the keystone of Bond's theory. Without the consideration of this variable, no sound comminution theory is possible.

After deducing the basis of the third theory, Bond states the third theory as follows:

"The total work useful in breakage which has been applied to a stated weight of homogeneous broken material is inversely proportional to the square root of the diameter of the product particles.

"Where K represents a proportionality constant and P is the size 80 per cent of the product passes, and is found from the plotted screen analysis,

$$W_t = K/\sqrt{P} \quad \text{.....(2*)}$$

"Equation (2) is the fundamental statement of the Third Theory".

If the rock is to be ground from an infinite size to a certain product size, it is obvious that, if the useful energy input in severance of the rock from its orebody varies, then the useful energy input in crushing and grinding to obtain the required product also varies, but the sum of useful energy inputs into this required product is always constant. This is expressed by Bond by his fundamental statement of the Third Theory.

*Numbers used for equations are from F. C. Bond's (3) original paper.

5.— *Bond's Work Index*

In introducing the Third Theory, Bond (3) makes the following statements regarding the requirements of a successful theory:

"(2) A 'successful theory' should be supported by a large mass of operating data on a wide variety of materials correlated with laboratory tests on the same materials.

"(3) The correlation should be made by a *work index* representing the total work input to a definite product size, which should be obtainable from any laboratory or commercial operation when the energy input and size analyses of feed and product are known.

" (4) It should be possible to calculate the useful energy input required for any size reduction from the work index or from any similar reduction in a different size range, when allowance is made for possible different breakage characteristics and machine efficiencies at different sizes".

Bond is defining the work index as follows:

"The work index (W_i) is the calculated KWH/ton applied in reducing material of infinite particle size to 80 per cent passing 100 microns, equivalent to approximately 65 per cent passing 200 mesh. It establishes the relative reduction resistance of a material in the size range tested, and the relative mechanical efficiencies of different machines and different processes. It can be found from any commercial laboratory operation where the work input and size distribution of the feed and product are known".

From his fundamental statement of the Third Theory, Bond derives the following expression for the work index:

$$W_i = W\frac{\sqrt{F}}{\sqrt{F}-\sqrt{P}}\sqrt{\frac{P}{100}} \quad \text{.....(5)}$$

where W = work or energy input in KWH/ton to a machine reducing material from a definite feed size (F) to a definite product size (P). The feed size, F, is the diameter in microns of the square hole which will pass 80 per cent of the feed and is found from the plotted screen analysis.

"When the work index is known, the energy input W required to break at the same efficiency from any feed size F to any product size P, in microns, is found from

$$W = W_i\frac{\sqrt{F}-\sqrt{P}}{\sqrt{F}}\sqrt{\frac{100}{P}} \quad \text{.....(6)}$$

"When F and P are in inches, the number 100 in equations (5) and (6) should be replaced with 0.003937, and $\sqrt{0.003937} = 0.0627$"

Bond says:

"If breakage characteristics of a material remain constant over all size ranges, and the mechanical efficiency of all machines employed is the same, then the values of the work index calculated under all different conditions should be constant. The work index expresses primarily the resistance of a material to breakage, and variations reveal differences in the breakage characteristics at different sizes and differences in the efficiencies of different machines and operations".

6.—*Work Index as a Function of Grindability*

The work index is evidently a function of grindability; hence, having defined the work index, it was logical to try and work out a formula expressing it in terms of grindability.

Taking a number of installations of wet overflow ball mills, 7½ feet in diameter inside the shell, operating in closed circuit with classifiers, and a number of wet overflow rod mills, 6 feet in diameter inside the shell and operating in open circuit, Bond (3) arrived at the following empirical equations:

$$W_p = 23/G_{rp}^{0.625} \quad \text{.....(13)}$$

$$W_p = 20/G_{bp}^{0.820} \quad \text{.....(14)}$$

where Gr_p and Gb_p are rod mill and ball mill grindabilities, respectively, at the product size P_1, and W_p is the KWH per ton of material passing the product size.

Applying the concepts of the Third Theory, Bond is expressing the work index, W_i, in terms of W_p by the following equation:

$$W_i = 0.80W_p\sqrt{P_1/100} \quad \text{.....(16)}$$

where P_1 is the sieve opening in microns at which grindability tests were made.

The coefficient $0.80\sqrt{P_1/100}$ is defined as the work index factor, which can be denoted as f. Thus, the equation (16) can be written as

$$W_i = fW_p \quad \ldots\ldots\ldots\ldots\ldots\ldots\ldots\ldots (16a)$$

7.—*Grindability as a Function of P ('80 per cent passing')*

The writer has taken at random, from Bond's (2) article, ball mill grindability figures for Malartic Gold Fields and Anaconda Copper Mining Company ores, and rod mill grindability figures for Lake Shore Mines ores, for each of which figures are available for the complete sequence of meshes. Assuming that '80 per cent passing', P, is the lower limit of the next mesh size (17), Tables I and II were obtained.

Plotting *log G* as ordinates against *log P* as abscissae, straight lines are obtained (Figures 1, 2, and 3). (*A*), (*B*), and (*C*) are the equations for these lines.

Substituting G in equations (13) and (14) by its values in equations (*A*), (*B*), and (*C*), and then substituting W_p in equation (16*a*) by values obtained in (13) and (14), the (*D*), (*E*), *and* (*F*) empirical equations are obtained.

TABLE I.—BOND'S BALL-MILL GRINDABILITY TESTS

Malartic Gold Fields Ore			Anaconda Copper Mining Co. Ore		
P_1, mesh	P, microns	G, gm. per rev.	P_1, mesh	P, microns	G, gm. per rev.
28	350	2.86	28	350	4.14
35	250	2.31	35	250	3.42
48	177	2.25	48	177	3.13
65	125	2.04	65	125	2.53
100	88	1.67	100	88	2.05
150	62	1.36	150	62	1.40
200	44	1.03	200	44	1.175

For M.G.F. ore:

$$G_{bp} = \frac{P^{0.493}}{5.854} \quad \ldots\ldots\ldots\ldots\ldots\ldots (A)$$

For A.C.M. ore:

$$G_{bp} = \frac{P^{0.636}}{9.184} \quad \ldots\ldots\ldots\ldots\ldots\ldots (B)$$

For L.S.M. ore:

$$G_{bp} = \frac{P^{0.627}}{8.318} \quad \ldots\ldots\ldots\ldots\ldots\ldots (C)$$

For M.G.F. ore:

$$W_i = 85.20f/P^{0.404} \quad \ldots\ldots\ldots\ldots\ldots\ldots (D)$$

For A.C.M. ore:

$$W_i = 123.22f/P^{0.522} \quad \ldots\ldots\ldots\ldots\ldots\ldots (E)$$

For L.S.M. ore:

$$W_i = 86.44f/P^{0.392} \quad \ldots\ldots\ldots\ldots\ldots\ldots (F)$$

TABLE II.— BOND'S ROD-MILL GRINDABILITY TESTS
LAKE SHORE MINES ORE

P_1, mesh	P, microns	G, gm. per rev.
6	2,000	13.1
8	1,410	11.28
10	1,000	9.3
14	710	7.36

Using values of f calculated by Bond (3), the following values for W_i, shown in Tables III, IV, and V, for grindabilities at given meshes are calculated from equations (*D*), (*E*), and (*F*). A fourth column containing values of W_i obtained by Bond (3) in his laboratory tests

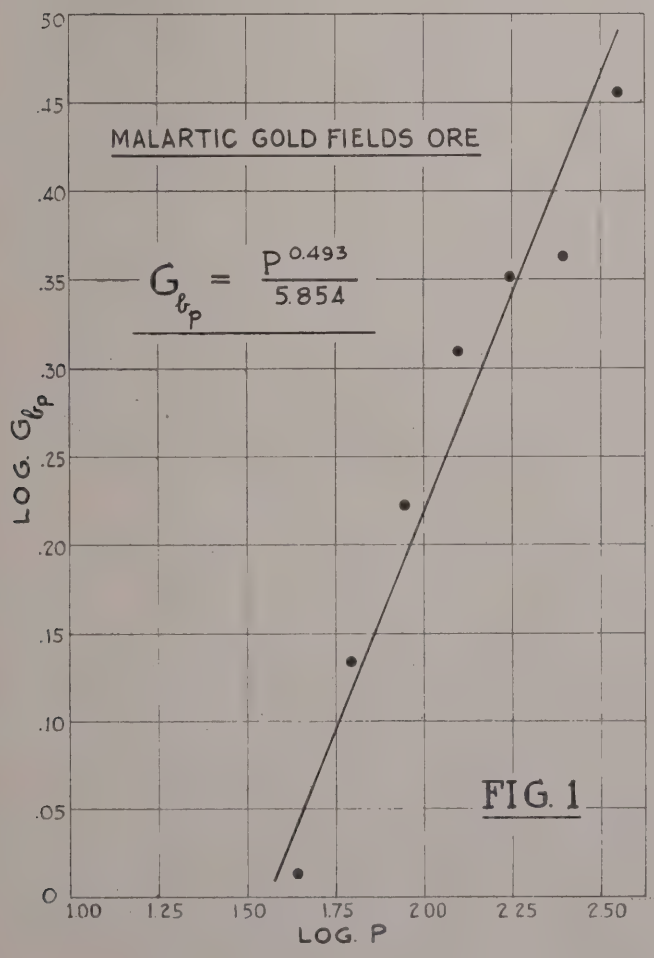

FIG. 1

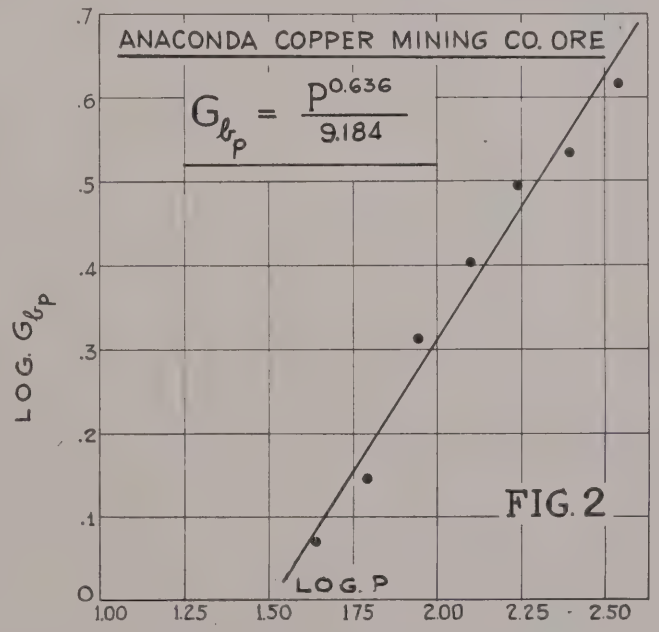

FIG. 2

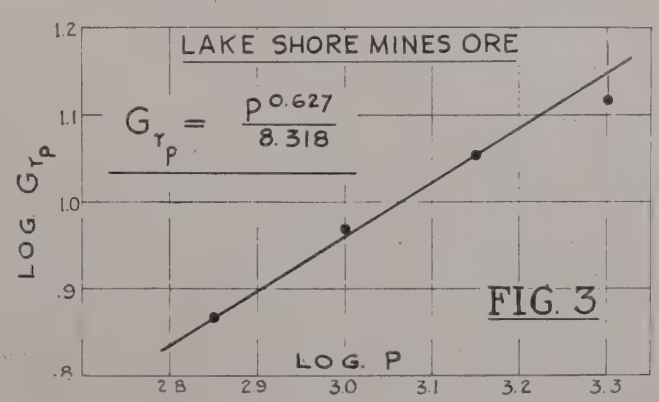

FIG. 3

TABLE III.— MALARTIC GOLD FIELDS ORE (B.M.)

P_1, mesh	P, microns	W_i (calc.)	W_i (Bond)
28	350	15.52	15.35
35	250	15.01	16.40
48	177	14.50	13.98
65	125	14.04	13.05
100	88	13.66	12.69
150	62	13.17	12.88
200	44	12.72	13.44

TABLE IV.— ANACONDA COPPER MINING CO. ORE (B.M.)

P_1, mesh	P, microns	W_i (calc.)	W_i (Bond)
28	350	11.24	12.65
35	250	11.31	12.20
48	177	11.39	10.97
65	125	11.46	10.80
100	88	11.65	11.81
150	62	11.71	12.88
200	44	11.77	12.04

TABLE V.— LAKE SHORE MINES ORE (R.M.)

P_1, mesh	P, microns	W_i (calc.)	W_i (Bond)
6	2,000	20.37	21.40
8	1,410	19.65	19.92
10	1,000	18.91	18.80
14	710	18.20	20.70–18.25

on the same ores is added to each table for purposes of comparison.

Discrepancies in the calculated values of W_i are probably due, at least partly, to the fact that '80 per cent passing' points have been assumed, instead of being determined by the screen analysis after each grindability test. On the other hand, when the calculated values of W_i are compared with those obtained by Bond (3) in his laboratory tests, the agreement is striking.

From the preceding discussion it would appear that grindability can be expressed in terms of the '80 per cent passing' point by the following general equation:

$$G = aP^1 \quad \text{(G)}$$

Bond has shown that

$$W_t = 0.80 W_p = K_1/G^m$$

By substituting G in this equation by aP^1,

$$W_t = K_1/aP^{1m} = K/P^n \quad \text{(2)}$$

which is Bond's fundamental statement of the Third Theory.

Empirically, the exponent n for the three ores (M.G.F., A.C.M., and L.S.M.) was found to be: 0.404, 0.522, and 0.392, respectively, which is in good agreement with Bond's theoretical value of $n = 1/2$.

8.—*Thermodynamic Criteria*

Bond (3), in agreement with the concepts of the Third Theory, defines the comminution process as follows:

> "The comminution process is a process in which mechanical kinetic energy is transformed into heat through internal and external friction, under conditions such that critical strains are exceeded and material is broken".

This definition suggests that principles of thermodynamics are involved in the Third Theory; hence it poses the question of thermodynamic efficiencies of the comminution process. Bond (3) defines mechanical efficiency of the comminution process as the ratio of useful work to work input. Does the mechanical efficiency correspond to the thermodynamic efficiencies?

Some evidence of this is found in the following statement by Taggart (18):

> "A large part of the energy input to the mill is converted to heat. Any departure from the temperature which, under a given set of conditions, corresponds to the desired grind, marks a change in the character of the grind".

In other words, if:

E_i = energy input
E_s = useful energy input (surface energy)
Q = heat generated

then

$$E_i = E_s + Q \quad \text{(H)}$$

If E_i is a constant, by equating it to unity we get

$$1 = E_s + Q \quad \textit{or} \quad \frac{E_s}{Q} = \frac{1}{Q} - 1$$

Denoting the ratio of E_s to Q by N,

$$E_s = NQ \quad \text{(I)}$$

The writer postulates that equation (*I*) is fundamental in grinding. E_s is a thermodynamic property which is expressed in energy units and is characteristic of each material, and coefficient N is postulated as a thermodynamic constant which is distinct for the grindability of each material under given conditions such as ball load, amount of feed to the mill, dilution, etc.

Since thermodynamic efficiency, e_t, is equal to the ratio of theoretical work (power) to actual work (power) input, where theoretical work can be equated to indicated or useful work input ($E_s = W_u$) and the actual work input to the kinetic energy input ($E_i = W_k$) into the grinding media,

$$e_t = \frac{E_s}{E_i} \times 100 = \frac{E_s}{E_s + Q} \times 100 \quad \text{(J)}$$

Substituting the value of E_s from equation (*I*) in equation (*J*),

$$e_t = \frac{E_s}{E_s + Q} = \frac{N}{1+N} \quad \text{(K)}$$

where N can be obtained directly from experimental data.

Evidently, the higher the value of N, the higher is the thermodynamic efficiency.

Recently, Schellinger (13), after a series of calorimetric determinations of surface-energy relationships for some common minerals, has postulated that, in grinding, 10 to 20 per cent of the energy input appears as the energy absorbed by the minerals present and the remaining 90 to 80 per cent appears as heat. Hence, from Schellinger's experiments, N in equation (*K*) varies from 10/90 to 20/80, or from 0.11 to 0.25.

If N can be determined, for instance, from Bond's grindability data, then the useful work input for a given ore can be found directly from equation (*K*).

Mechanical efficiency, e_m, as definded by Bond (3), $W_u/W = e_m$, can be obtained in the following manner:

If W_f is the power loss due to friction,

then

$$W_k + W_f = W = \text{energy supplied to motor}$$

and

$$\frac{W_u}{W} = \frac{W_u}{W_f} = e_m$$

from which

$$e_m\left(1 + \frac{W_f}{W_k}\right) = \frac{W_u}{W_k} = \frac{E_s}{E_i} = e_t$$

or

$$e_m = \frac{e_t W_k}{W} \quad \text{(L)}$$

Substituting e_t from equation (*K*),

$$e_m = \frac{NW_k}{(1+N)W} = \frac{N(W - W_f)}{(1+N)W} \quad \text{(M)}$$

Thus, mechanical efficiency is equal to the ratio of the product of N and kinetic energy input into the grinding media to the product of $(1 + N)$ and total energy supplied to the motor, or, assuming energy input to the motor to be constant, e_m is a function of the thermodynamic constant of the material, N, and power loss due to friction.

Conclusion

In conclusion, the writer would like to advance the following idea, using it as a heuristic principle:

If the equation $E_s = NQ$ is fundamental in grinding, then, in order to grind, heat has to be generated in the process. Hence, a cylindrical mill is probably a very efficient machine if, in it, only that part of the input energy is degraded to heat which is absolutely essential for the required comminution. Extending further this idea, it is just possible that if the waste heat from compressors, waste steam from boilers, etc., is used to heat the ore or water input to the ball mill, a higher part of the energy input will go into actual grinding of ore, thus increasing the efficiency of the mill or its capacity.

This appears to be reasonable with such heterogeneous material as ores, expansion of which with the application of heat will weaken the adhesion between constituent minerals and generate stresses between cleavage planes of individual crystals. Hence, it is suggested that at least a part of that amount of input energy which is converted to heat to do the work of generating stresses between the boundaries of minerals can be replaced by external heat and thus diverted into straight comminution.

Would it be far-fetched to ask that, if this principle works, both the capital cost of grinding plants and the cost of their operation will come down? The essential point is that the unit of external heat must be applied at a cost lower than the cost of the unit heat generated by the action of the mill.

Acknowledgments

The idea of this paper was born as a result of the writer's conversations with Mr. A. L. Blomfield, Chairman of the Board, Lake Shore Mines. The contents of the paper were stimulated by *The Third Theory of Comminution* of Mr. F. C. Bond, Engineer-in-Charge of Technical Research, Allis-Chalmers Manufacturing Company. With both of them the writer has been in close correspondence and, when viewed in retrospect, several ideas, incorporated in this paper, were the outcome of this correspondence.

To these two faithful engineers, Mr. Blomfield and Mr. Bond, this paper is gratefully dedicated.

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DISCUSSION

Mr. Fred C. Bond (Engineer-in-Charge, Technical Research, Allis-Chalmers Manufacturing Company, Milwaukee, Wis.): Mr. Djingheuzian's favourable comments regarding the Third Theory of Comminution are greatly appreciated. The Third Theory furnishes a method by which the power consumption of new crushing and grinding installations can be anticipated, and the relative efficiencies of different machines with different feed and product sizes can be calculated and compared with more accuracy than has heretofore been possible. By calculating the work index values, information can be gained regarding the effects of changes in the circulating load, pulp dilution, ball and rod sizes, mill speed, mill diameter, lifter type, and all other important operating variables; and the most efficient operating conditions can be selected.

When the reduction ratio Rr equals the '80 per cent passing' feed size over the '80 per cent passing' product size, or F/P, the ratio of the work index W_i to the work input W can be found by the following equation

$$\frac{W_i}{W} = \left(\frac{\sqrt{R_r}}{\sqrt{R_r - 1}}\right)\sqrt{\frac{P}{100}}$$

When W_i/W is plotted against P on log-log paper, straight lines can be drawn for all desired Rr values in the above equation, and this plot can be used to find W_i/W for any feed and product size, thus making individual calculations unnecessary.

However, the Third Theory in its present form indicates only relative efficiencies, and does not determine the absolute mechanical efficiency of any comminution operation. Mr. Djingheuzian has approached this problem from the thermodynamic viewpoint, and has logically developed the assumption that the ratio of the work input transformed into new surface energy to that transformed into heat is constant for each material. In the writer's opinion, the validity of this assumption is the question of greatest immediate importance in the development of comminution as a science. A large amount of test work and theoretical reasoning will be necessary to establish or disprove it.

The Third Theory indicates that the crack length, rather than the new surface area, is the proper criterion of the work done in crushing and grinding; the crack length is proportional to the square root of the new surface area.

The surface area assumption is logically approached from the thermodynamic viewpoint, while the crack length assumption is approached from the physical properties viewpoint — including the compressive strength and the modulus of elasticity. Considerable development along both lines is possible and desirable.

The absolute efficiency of comminution is a problem of great importance which will eventually be solved. Mr. Djingheuzian's present paper and his future work in this field will aid in solving the problem and should stimulate the interest of other investigators.

Mr. H. R. Banks (Superintendent, Sullivan Concentrator, The Consolidated Mining and Smelting Company, Limited, Chapman Camp, B.C.): Salutations to the author of this paper, who continues to spur us on along the path of 'exploration and underlying fundamental laws of comminution'. Messrs. Bond and Maxson very wisely stated some thirteen years ago that such procedure would make the greatest contribution to the advances in our knowledge of this most necessary tool in industry.

Bond, in his 'Third Theory of Comminution', has given us much finer and more dependable divisions than we had previously on the scale used to measure work being done. By the same token, he has also provided us with a scale to measure the work we have not done!

Mr. Djingheuzian, in his paper, introduces a very definite approach to one of the fundamentals in comminution. I believe that the application of heat, either in the process of breaking down the particle or by extraneous means, lowers cohesive forces in the material being broken down by reducing the elastic limit of those particles.

It has always been my concept that the richest field for investigation of fundamental principles of comminution lies in a study of that energy absorbed within the elastic limit of the material to be ground.

With Bond's Third Theory now available and apparently accepted — since there has been little written in opposition — the measurement of effective energy is more readily discernible.

We do not agree with Mr. Djingheuzian in his conclusion that the idea submitted is entirely a heuristic principle. I believe that the principle cited is demonstrable in fact.

I would like to make the suggestion that our Institute approach the parallel institutes in England, South Africa, Australia, and the United States with the object of setting up a world-wide committee for the investigation of the fundamental principles of comminution.

In each of the countries mentioned there has been some excellent work done and, by co-ordinating and sorting relevant hypotheses under the guidance of such a world-wide committee, I believe that great advances should be achieved. One function of such a committee would be to formulate a plan of approaching the problems in a more orderly manner than we are now doing.

Mr. A. L. Blomfield (Chairman of the Board, Lake Shore Mines, Limited, Kirkland Lake, Ont.): The author offers for discussion an analysis of recent advances made in grinding. Following in the footsteps of Bond's *The Third Theory of Comminution*, this paper is welcome.

Having submitted the studies of several investigators as grinding criteria, the author has made an attempt to develop the thermodynamic implications of the Third Theory. This is opening a new avenue for grinding investigators.

To me, the most important criterion discussed by the author is the '80 per cent point'. Bond's work index, provided the basis of sizing is standardized, gives us means to getting to the comparative efficiencies of our grinding plants. However, to do so F and P must be strictly defined. So what is the '80 per cent passing'? How near does it approach to being a scientific term?

The author mentions the Lake Shore measure of grind, *i.e.*, plus 28 microns, which has varied from 15 to 30 per cent, and submits it as a proof of 80 per cent point.

During fine-grinding investigations at Lake Shore, as I saw this, it was not the 80 per cent point, but it was set at the nearest figure safely eliminating the 'classification failure' to make a clean separation. As I recall, we chose 28 microns as a measure of our grind because, firstly, it was the strongest point in the efficiency of the infrasizer, and secondly, it was low enough down the sizer's results to be entirely free of classifier irregularities. At this fineness of grind, the bowl classifiers' performance at Lake Shore was approaching 100 per cent efficiency.

We also studied at the time Hollinger's results and thought that their use of 65 per cent minus 200 mesh was correct. Their classification was done by an old classifier, less efficient than newer machines, and so 80 per cent might not have fit into their requirements.

Granted that the 80 per cent figure eliminates the 'classification failure' in most cases. However, it appears to me that 80 per cent might be a generalization rather than a scientific statement. It seems to me that, only when we consider this point as a point where it becomes the strongest point in sizing and low enough down the sizer's results to be entirely free of classifier irregularities, does it become a scientific term.

At the time, we tied plus 28 microns at Lake Shore into 'constant fractions'. These 'constant fractions' were obtained at Lake Shore both in the plant and in the laboratory. I have practically no doubt that grinding of any material can be tied into 'constant fractions'. I believe the author has also proved the existence of 'constant fractions' at Siscoe. So it appears to me that 'percent passing' should be made a function of 'constant fractions'. This I offer as a field of investigation to the author who, I understand, is now carrying on a research on the influence of application of external heat on the efficiency of grinding. The suggested investigation on 'constant fractions', I believe, will fit in nicely with his present experiments.

Mr. Van H. Smith (Manager, Canadian Division, North American Cyanamid, Limited, Toronto): It has occurred to me that the higher efficiency in grinding, resulting when higher temperatures are present, may result from a change in viscosity of the pulp.

I think that, in Mr. Djingheuzian's formulae, a viscosity variable has to be taken into consideration; also that, what may be correct in a test unit may not apply in a full-scale grinding unit.

I suggest that a study be made of the drillability of known and unknown ores, using same diameter drill bit, same strokes per minute, and same air pressure, and that, from the results of such a study, an attempt be made to tie down whether grindability in a ball mill can be roughly measured by the drilling characteristics obtained under such similar test conditions.

I know that the two problems are different, but I feel that they may be closely enough related to give valuable information, or perhaps correlation of data obtained in other ways.

Mr. H. M. Wright (President, Wright Engineers, Limited, Vancouver): In commenting on Mr. Djingheuzian's paper, I would first like to thank him on behalf of the Metallurgy Division for his very fine contribution to this Winnipeg Meeting. Mr. Djingheuzian has done a vast amount of work in preparing his paper and it is certainly a very worthwhile contribution to our Institute. I would also like to thank Mr. Bond, of Milwaukee, for his co-operation in coming to this meeting personally.

Some excellent discussion has developed from this paper this afternoon and in addition to this, of course, we have the very fine contributions by Mr. Banks and Mr. Blomfield. This paper will be published in the Institute Bulletin shortly and we plan on having a further discussion period on this subject at the Edmonton Meeting next April.

Mr. Djingheuzian has discussed heat generation and has indicated that test work shows grinding improves with heat. I wonder if his experiments could be carried still further by attempting grinding under 'refrigerated' conditions. Possibly, the grinding equipment, as well as the ore charge, could be refrigerated and the tests carried out under refrigerated conditions, say at 32 degrees. It would be interesting to see the results from a series of grinding tests with increases in temperature from 32 degrees in increments of 10 degrees.

To my knowledge, inferior flotation results under winter conditions have usually been attributed to cold water, which in turn has affected reagent activity. Possibly, these cold-weather conditions have also affected grinding and this may be partially the cause for a variation in flotation results.

Author's Reply to Discussion

I wish to express my appreciation and gratitude to all who have so ably contributed to the discussion. Their comments and criticisms, complementing the paper to such a high degree, show what a live subject we have on our hands once we start talking about grinding.

Mr. Blomfield's thoughtful remarks regarding the '80 per cent point' are a significant contribution to the understanding of this point as a measure of grind by every operator. Mr. Blomfield's definition, which goes back fifteen years, agrees fundamentally, in my opinion, with Mr. Bond's (1). However, it goes to the very heart of the operator's grinding problems and thus makes the '80 per cent point' a concrete scientific term for him.

Mr. Blomfield has also remarked on 'constant fractions'. This introduces another highly important grinding criterion. 'Constant fractions' undoubtedly are a function of the petrological nature of the rock. Following Mr. Blomfield's contribution, I have looked into some of my laboratory grinding results, and, so far, it would appear that 'constant fractions' are a fundamental characteristic of each material tested. Fortunately for me, the investigation on 'constant fractions', suggested by Mr. Blomfield, would fit in nicely with my present investigations. It would be very interesting if operators would analyse their plant results for 'constant fractions'.

Mr. Bond, in his contribution, postulates two assumptions: the surface area and the crack length. The surface area assumption means thermodynamic approach, hence a correct understanding of the term 'surface energy'. So what, exactly is 'surface energy'? At present, I would like to submit that it is the amount of useful work, say in ergs, which goes into making of one sq. cm. of new surface. Evidently, the validity of the assumption that N is a constant will depend on the correct definition of 'surface energy'. To my mind, the key to this definition lies in finding out, at least theoretically, what will happen if no heat is dissipated from the grinding machine, and the process thus becomes an adiabatic one. At the moment, I can only imagine simple implications that the solution of this problem will entail.

The other assumption is the crack length assumption. It is approached from the physical properties viewpoint, including the compressive strength and the modulus of elasticity. This concept undoubtedly offers great possibilities and throws another door wide open to the investigator.

A third concept is offered by Mr. Banks, namely, that of the elastic limit as a criterion in the study of the energy absorbed in the useful work in comminution. My reaction to Mr. Banks' postulate is that his concept might finally lead us to properly quantifying thermodynamic efficiencies in comminution, and hence mechanical efficiencies, as my equation (M) indicates.

I do not apologize for using 'heuristic principle'. Webster's New Collegiate dictionary defines 'heuristic' as "serving to discover or to stimulate investigation", and it is in this sense that I have used the word. As far as I know there is no indication in the literature that anybody has ever investigated this principle. It was when I read Bond's definition of comminution that I began to see the light. Naturally at the first chance after postulating the principle as a 'heuristic' one, I have gone into the laboratory, have so far carried out four series of experiments, and they all have confirmed the principle, thus eliminating 'heuristic'. To my mind, this has been the best proof of Bond's definition of comminution.

Mr. Van Smith's comment that the higher efficiency in grinding resulting when higher temperatures are present may result from a change in viscosity of the pulp deserves utmost attention. If by higher efficiency is meant an increase in surface produced, then I do not think that viscosity has anything to do with it. On the other hand, if by higher efficiency is meant that less power might be consumed, then viscosity of the pulp might have something to do with the increased efficiency. At present, I have not sufficient evidence to answer Mr. Van Smith conclusively. However, I hope that at a later date I will have sufficient data which might explain the influence of viscosity on the efficiency of grinding.

In this connection, during the discussion, one of the Cominco engineers, Mr. J. F. Buckingham, asked whether the application of external heat would also improve the effi-

(1) Bond, F. C., *Mathematics of Crushing and Grinding;* Proceedings, Symposium on Mineral Dressing, I.M.M., September, 1952.

ciency of dry grinding? I think it would, since the same principle of particle expansion that applies in wet grinding would also apply in dry grinding: the ore has to be heated before it is fed into the grinding machine.

Mr. Van Smith's suggestion regarding a study of the drillability of known and unknown ores is an exceptionally interesting one. I hope that studies will be undertaken along the lines suggested. When the results of such studies are compared to Mr. Bond's grindability results, I have very little doubt that some very important correlations can be made.

Mr. Wright has made a very good suggestion with respect to the experiments recently started at the Mines Branch. The last paragraph in his contribution, it seems to me, deserves the particular attention of all mill superintendents operating flotation mills under our severe winter conditions. It just goes to show how many ramifications there are in such a grinding research, and, still more, it shows in a concrete manner that the development of the science of grinding is not a one man's job but a team's. Only by team work, I believe, will we finally get there: It is the flow of suggestions and reporting on any pertinent facts that will finally lead us to establishing grinding on a fundamental basis.

Oil From Coal*

By T. E. WARREN† and K. W. BOWLES†

(*Ottawa General Meeting, January,* 1952)

(*Transactions, Volume LV,* 1952, pp. 383-386)

Introduction

THE PROBLEM of producing synthetic liquid fuels from coal and other raw materials is of increasing importance in Canada as well as in the United States. This is due, to a large extent, to the continually increasing demand for gasoline and other liquid products which are now obtained solely from petroleum. The United States has reached a position where domestic supplies of crude petroleum must be augmented by imports and Canada is still a long way from self-sufficiency. In the event of war, there might well be an immediate need to find supplementary sources of liquid fuels. It is, therefore, expedient to consider alternative sources of production now — well in advance of the time when supplies of petroleum may prove inadequate to meet requirements.

The purpose of the present paper is to give a short review of the history of liquid fuel production from coal, to describe briefly the research sponsored by the Federal Government on the coal hydrogenation process, and finally to evaluate the present status and future trend of the synthetic fuel industry, with special reference to necessary research work.

History of the Oil-from-Coal Industry

The history of coal liquefaction began in 1913, in Germany, with the work of Frederick Bergius. His first approach to the subject was not deliberately planned. He was studying the reactions involved in the conversion of vegetable matter to coal and in the course of his experiments he made a sample of artificial coal. His ideas about its chemical composition suggested that it might react with hydrogen and so he was led to an experiment in which he treated the artificial coal with hydrogen at a temperature of 750°F. and a pressure of about 2,000 pounds per square inch. He found that the artificial coal was converted to a heavy oil and, when he repeated the experiment with natural coal, he obtained a similar result.

During the first World War, Bergius was encouraged to develop his discovery into a large-scale process. He finally built a small pilot plant which converted about one ton of coal per day to heavy oil. In 1923, I. G. Farbenindustrie took over the development of the process and in 1927 that company erected a full-scale hydrogenation plant at Leuna which converted brown coal to gasoline. Basically, this process consisted of reacting the coal with hydrogen by means of elevated temperature and pressure and a suitable catalyst.

In England during the 1930s, Imperial Chemical Industries developed hydrogenation to the point where it could be applied on a full scale to bituminous coal. The Company made an arrangement with the government of the United Kingdom for a tariff preference and erected a plant at Billingham for the conversion of bituminous coal and coal tar to gasoline.

The Billingham plant was well in advance of the German technique at that time and an almost identical copy of it was built at Scholven, in the Ruhr coal field. The Germans then possessed techniques for conversion of both brown and bituminous coals to liquid fuels and, utilizing this knowledge, they erected twelve major coal hydrogenation plants.

About 1930, an entirely different method was developed, in Germany, by Franz Fischer and Hans Tropsch. It is known as the *Fischer-Tropsch* or *gas-synthesis* process. In this method, water gas is made from coal and the hydrogen and carbon monoxide mixture is then catalytically converted to liquid hydrocarbons. Industrial development of this process was undertaken by Ruhrchemie, which erected nine plants in Germany between 1933 and 1939. In this process, about 1,000 cubic feet of water gas are required to make one gallon of liquid product. In Germany, most of the water gas was made from coke in Humphreys-Glasgow generators.

By 1943, the combined production from hydrogenation and Fischer-Tropsch plants was about 70 per cent of the total German oil consumption. The hydrogenation process made all of the aviation gasoline; the Fischer-Tropsch or gas-synthesis process played only a minor part, contributing about one-eighth of the total supply. It was recognized in Germany that hydrogenation was the better process for production of liquid fuels. Consequently, by the end of the War, more hydrogenation plants were being built, while construction of Fischer-Tropsch plants had ceased in 1939. As the synthetic oil industry was essential to the German war effort, all of the plants were heavily bombed and put out of operation by the Allied Air Forces.

During the war years, the United States Bureau of Mines had been preparing to expand its activities in the field of synthetic fuels production. It was clearly realized by the government that the industrial and social structure of the country was dependent on liquid fuels, and the uncertainty of the future supply of petroleum made it necessary to prepare for production of oil from other source materials. In 1944 Congress voted 30 million dollars for this purpose. After a thorough study of the German industry, the Bureau of

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†Engineers Fuels Division, Mines Branch, Department of Mines and Technical Surveys.

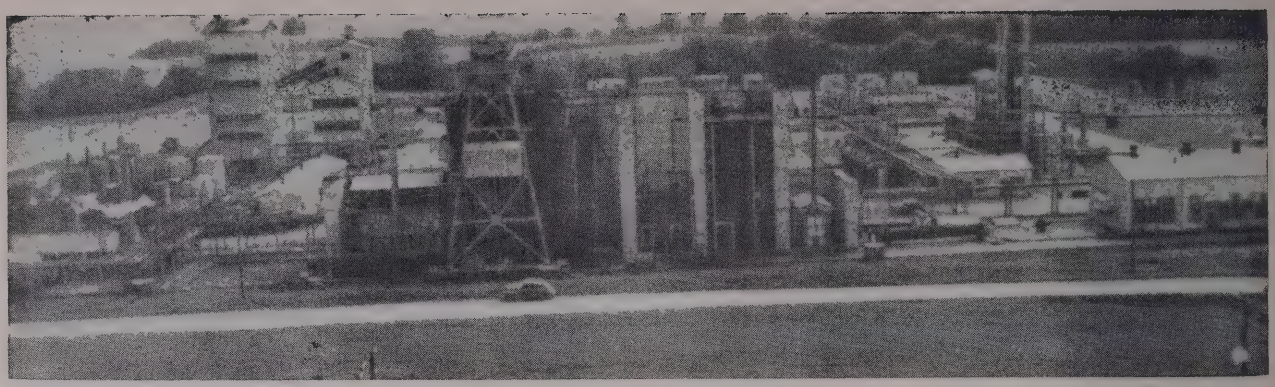

Figure 1.—U.S. Bureau of Mines coal hydrogenation demonstration plant at Louisiana, Missouri.
Coal and coal paste preparation units at left, high-pressure stalls in centre, refinery and light-oil storage at right.

Mines proceeded to erect laboratories and pilot plants for investigation of methods for producing synthetic liquid fuels. A large, well-equipped laboratory was built at Bruceton, Pennsylvania, near Pittsburgh, to study the fundamental chemistry of both the hydrogenation and Fischer-Tropsch processes. Two large-scale demonstration plants, one for each of the major processes, were also erected at Louisiana, Missouri. Figure 1 shows the hydrogenation plant and Figure 2 the Fischer-Tropsch plant. Laboratory and pilot-plant equipment were also installed at Morgantown, West Virginia, for studying methods of gasifying coal for the production of synthesis gas for the Fischer-Tropsch process. In addition to these three institutions, which are primarily for the study of the conversion of coal to oil, other facilities were built to investigate production of liquid fuels from oil shale. This investigation included the study of methods of mining and extracting the oil shale at Rifle, Colorado, and research on the refining of the shale oil at Laramie, Wyoming.

Two additions to the Synthetic Liquid Fuels Act have increased the total authorized expenditure to $87,600,000 and extended the investigational period to 1955. Recently, it has been proposed that full-scale commercial plants be built as a final step in the development programme, but no action has been taken yet in this respect.

As a result of this concentrated expenditure of money and effort, the U.S. Bureau of Mines has obtained a great deal of valuable technical information regarding the production of synthetic fuels, and both the hydrogenation and the Fischer-Tropsch processes as applied to coal have been improved in comparison with the original German procedures. In the hydrogenation process, the overall thermal efficiency has been increased from about 30 per cent to 45 or 50 per cent. In the Fischer-Tropsch process, a more active catalyst and an improved method of controlling the reaction temperature have been developed.

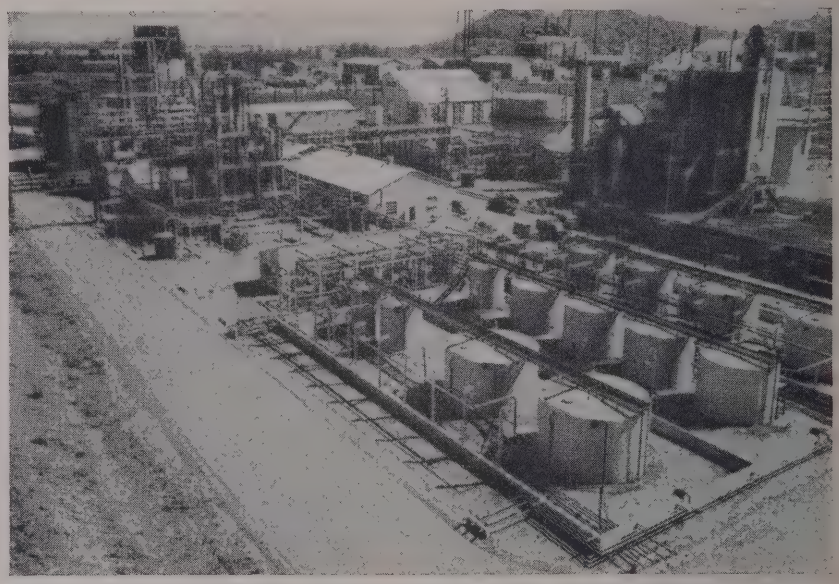

Figure 2.—General view of U.S. Bureau of Mines Fischer-Tropsch demonstration plant at Louisiana, Missouri.

Research Work of Mines Branch

Research on synthetic liquid fuels has been undertaken since 1930 by the Mines Branch of the Department of Mines and Technical Surveys at Ottawa. This work has been mainly devoted to the investigation of the hydrogenation process as applied to the fuel resources of Canada and has dealt particularly with the hydrogenation of coal. Before the last war, an assay was made of the amenability of Canadian coals to hydrogenation (1). Representative coals, ranging in rank from medium-volatile bituminous to peat, were tested in a large laboratory-size liquid-phase plant and the results showed that most of the coals were suitable as raw material for the process.

During the war, work on hydrogenation was suspended to make way for more urgent projects, but on the close of hostilities research was resumed and has continued to the present. One of the new projects is to study the effect of higher pressures on the efficiency of the process. Experiments will be carried out in a small pilot plant suitable for continuous operation at pressures up to a maximum of 20,000 pounds per square inch. Part of the equipment has already been obtained and the remainder is being manufactured.

The utilization of higher pressures would at first appear to be a retrograde step from the point of view of reducing costs, but a more careful study indicates that it may have

(1) Warren, T. E., and Bowles, K. W., *Tests on the Liquefaction of Canadian Coals by Hydrogenation;* Bur. of Mines, Dept. of Mines & Resources, Ottawa, Rept. No. 798, 1940.

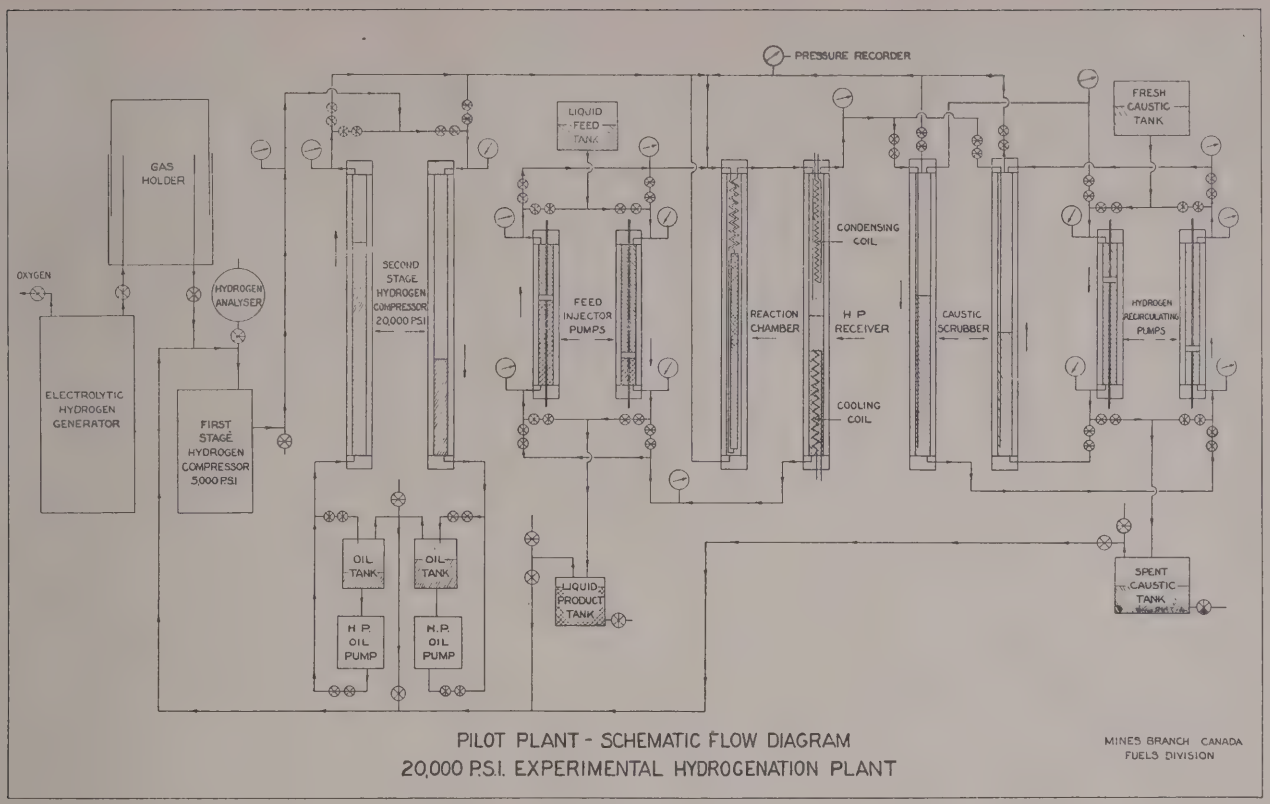

Figure 3.

practical possibilities. Considering the past history of the process, Bergius' first experiments were conducted at a pressure of about 2,000 pounds per square inch, the Leuna plant operated at 3,000, the Billingham plant at 5,000, and the final German plants at 10,000. With each increase in pressure there was improved economy. Furthermore, the itemized costs of the German plants showed that less than one-third of the total cost was incurred in operations involving the use of high pressures. For example, a typical German plant consisted of the hydrogen plant, the hydrogenation plant, the gasoline refinery, and the power plant. The hydrogenation section was the only one that utilized high-pressure equipment and, furthermore, much of this section was composed of ordinary structural and low-pressure equipment. The cost of increasing the pressure applies, therefore, to only a small portion of the plant. It is hoped that increased efficiency resulting from the use of higher pressures will reduce the overall costs of the process.

A detailed description of the process and the equipment is beyond the scope of the present paper, but the more important features are shown in the flow-sheet of the process in Figure 3. The hydrogen is compressed in two stages: first, by a compressor and, second, by the use of oil pumps. Compressed hydrogen, feed stock, and catalyst pass into the reaction chamber, where reaction takes place at the operating temperature and pressure. The products are cooled, the liquid portion being collected in the receiver. The uncondensed reaction gases and excess hydrogen pass through the caustic scrubbers, where hydrogen sulphide is extracted, and they are then recirculated through the reaction chamber.

Essential requirements are, first, that the plant will function satisfactorily under the exceptionally high operating pressures, and second, that uniform rates of flow of liquid feed and hydrogen be maintained through the reaction zone for relatively long periods of time. These requirements have made it necessary to design apparatus with novel mechanical features. One of the features is that ordinary pumps are not used for charging the feed into the high-pressure system. Upward movement of a piston in the feed injector charges the feed stock and at the same time withdraws an equal volume of the liquid product from the high-pressure receiver. While one injector is performing this operation, the other is discharging the liquid product and at the same time is being filled with fresh feed. This procedure gives accurate control of feed rate, it eliminates the need for liquid level measurement in the receiver, and it prevents surges and valving difficulties due to intermittent withdrawal of the liquid product. Hydrogen re-circulating pumps are also eliminated. The hydrogen is simultaneously re-circulated and scrubbed by caustic solution using similar injectors, which operate in the same manner as the feed injectors. If this method is used in a commercial-scale plant, the saving in power will be significant.

In the experimental programme, it is proposed, at first, to use oil extracted from the Alberta bituminous sands. This work will be followed by tests on domestic coals. It is hoped that the results of this project will contribute toward reducing the cost of the hydrogenation process.

Present Status and Future Outlook

In attempting to assess the present world status of the oil-from-coal industry, one conclusion is that the industry appears to be in a transitional period in which it can be established only under special circumstances. The production of valuable chemicals as by-products from the process will provide a limited stimulus. On the other hand, these products may comprise the primary basis of the operation. For instance, the Carbide and Carbon Chemicals

Company has built and is now operating a coal hydrogenation demonstration plant at Institute, West Virginia, for the production of aromatic chemicals. The cost of this plant, with a capacity of 300 tons of coal per day, was eleven million dollars.

Unusual geographical situations will also play a part. In this regard it is interesting to note that the Anglo Transvaal Consolidated Investment Company is building a coal hydrogenation plant in South Africa.

National defense is an important factor. In the event of war, the requirements for liquid fuels would increase tremendously while imports of petroleum might be very limited or cut off entirely. In such circumstances, the United States and Canada may be forced to resort to supplementary supplies of liquid fuels produced from coal or other domestic raw materials.

But the establishment of the oil-from-coal industry on a permanent basis will depend finally upon reaching a competitive position as a liquid fuel producer. It is not possible to predict, with any reasonable accuracy, when it will reach this position. The factors involved can be assessed only in a qualitative way. First, it is readily apparent from the relative magnitudes of the coal and oil reserves and from the relative rates of production that petroleum will be exhausted long before coal. Second, the cost of discovery and production of petroleum will tend to increase. Third, with further research the cost of production of oil from coal will be reduced. Finally, other source materials, such as bituminous sand and oil shale, will probably enter the competition.

The extent to which these other materials will influence the development cannot now be determined. But it should be emphasized that this last factor is of vital importance to the oil-from-coal industry, especially on this continent, and merits special study. In this regard it should be noted that Canada possesses an important substitute source of liquid fuels in the huge deposits of bituminous sands occurring in northern Alberta. As the separated oil from these tar sands requires less processing to produce fuels than does coal, it will probably be a more economical source than coal.

In view of the future possibilities of the demand and supply situation regarding liquid fuels, it is highly desirable that the necessary research on reducing the cost of synthetic fuel production should be continued during this transitional period. Canadian participation in the research cannot include a comprehensive development project similar to that of the United States Bureau of Mines because of the very high cost involved. With our limited budget we must, therefore, concentrate on a comparatively narrow front of the investigational field and especially on phases of fundamental research that are not included in the programme of the United States.

One such phase of research work that comes within the above category is the investigation of the influence of higher pressures on the hydrogenation of coal. As described above, this work is one of the research projects that is at present being undertaken by the Mines Branch at Ottawa.

The commencement of commercial production of liquid fuels from coal or other substitute material will take place almost certainly long before the oil reserves are exhausted. As a result of the unprecedented demand for petroleum products, the United States must now augment her domestic supply of crude oil by imports and, in spite of the increasing production of petroleum in Alberta and Saskatchewan, Canada can only supply a portion of her total requirements. Therefore, it is reasonable to assume that, in order to meet the increased domestic demands of the future, both Canada and the United States may have to supplement their present sources of supply by synthetic liquid fuels. Furthermore, the establishment of commercial production may take place much sooner in the event of a national emergency when adequate supplies will be critical.

Discussion

Dr. Montgomery: It occurred to me that many members of the audience might be interested to know how closely the chemical composition of the gasoline derived from coal compares with that currently produced from crude oil. I would therefore like to ask Dr. Warren if he would enlarge on this point.

Dr. Warren: The quality of the gasoline depends very largely on the final stages of refining. Operating conditions can be varied so as to produce a wide range in quality. Usually the gasoline made from coal is very similar to ordinary gasoline produced from petroleum. As an illustration of this point, samples of gasoline were obtained from captured German planes during the war and it was difficult to determine their origin from inspection of their physical and chemical properties. Gasolines are mixtures of paraffinic, olefinic, naphthenic, and aromatic hydrocarbons. Ordinarily, the gasoline produced from coal has a higher proportion of the aromatic constituents and a lower proportion of olefins. The overall conclusion is that ordinary gasoline can be made from coal and that there is no difficulty in that regard.

Mr. Burrough: I would ask Dr. Warren to discuss some of the figures on the cost of the process.

Dr. Warren: The cost figures that I have are not final and I decided not to give any cost figures at all in this paper. Anyone who would like to read something of the costs is referred to United States Bureau of Mines Report of Investigations Number 4564, where costs of producing gasoline from coal have been worked out in detail for a number of different operating conditions. The costs derived in this report are, however, much lower than those prepared by the National Petroleum Council's Committee on Synthetic Liquid Production Costs, and both statements should be taken into consideration. At present they are so far apart that I do not believe we are justified in making any statement about the cost of producing oil from coal.

A Reservoir Study of the Pincher Creek Field

*by F. D. AARING**

(*Petroleum and Natural Gas Division, Edmonton, May,* 1952)

(*Transactions, Volume LV,* 1952, pp. 387-395)

Introduction

THE PINCHER CREEK gas condensate field, in Township 3, Range 28, and Townships 3 and 4, Range 29, West of the Fourth Meridian, was discovered in 1947 by the Canadian Gulf Oil Company with the completion of Pincher Creek No. 1 well in the Madison limestone of Mississippian age.

This discovery was the result of a great deal of detailed seismic work in the area and is a credit to the geophysical personnel of the Company. The depth of the reservoir varies from 7,400 feet to 8,200 feet subsea, with the established water level at −8,200 feet. The predicted depth was but a scant 100 feet from that found in the discovery well, at 11,705 feet. Since the discovery of the reservoir, three additional wells have penetrated the Madison limestone and have further verified the original seismic interpretation of the limits of this reservoir.

To date, only a very small amount of gas has been produced from this field. Consequently, the reservoir studies that have been made so far are predominantly based on investigations of the phase behaviour of the fluid and physical characteristics of the reservoir rock. In view of the above limitations, it has been necessary to set up and perform model reservoir studies with the fluids and to recover as much core from the productive Madison limestone as possible in order that it might be analysed by conventional laboratory technique and be subjected to detailed microscopic geological examination.

Geological Description

Figure No. 1 is a stratigraphic and structural column of the formations encountered in Marr No. 1 while drilling to the Mississippian. The first strata encountered below the glacial drift are shales of the Colorado formation of Upper Cretaceous age. These shales are underlain by a series of thrust-faulted repetitions of Colorado (Upper Cretaceous) and Blairmore (Lower Cretaceous) sections to a depth of 11,586 feet.

The Colorado formation is composed chiefly of black fissile shale of marine origin, locally showing calcareous specks. Siltstone and sandstone streaks are common. This is underlain by the Blairmore formation of Lower Cretaceous age.

The Lower Cretaceous is divided into an upper non-marine shale, sandstone, and conglomerate (Blairmore), and an underlying Kootenay formation composed of coal, carbonaceous sandstone, and basal brown sandstone.

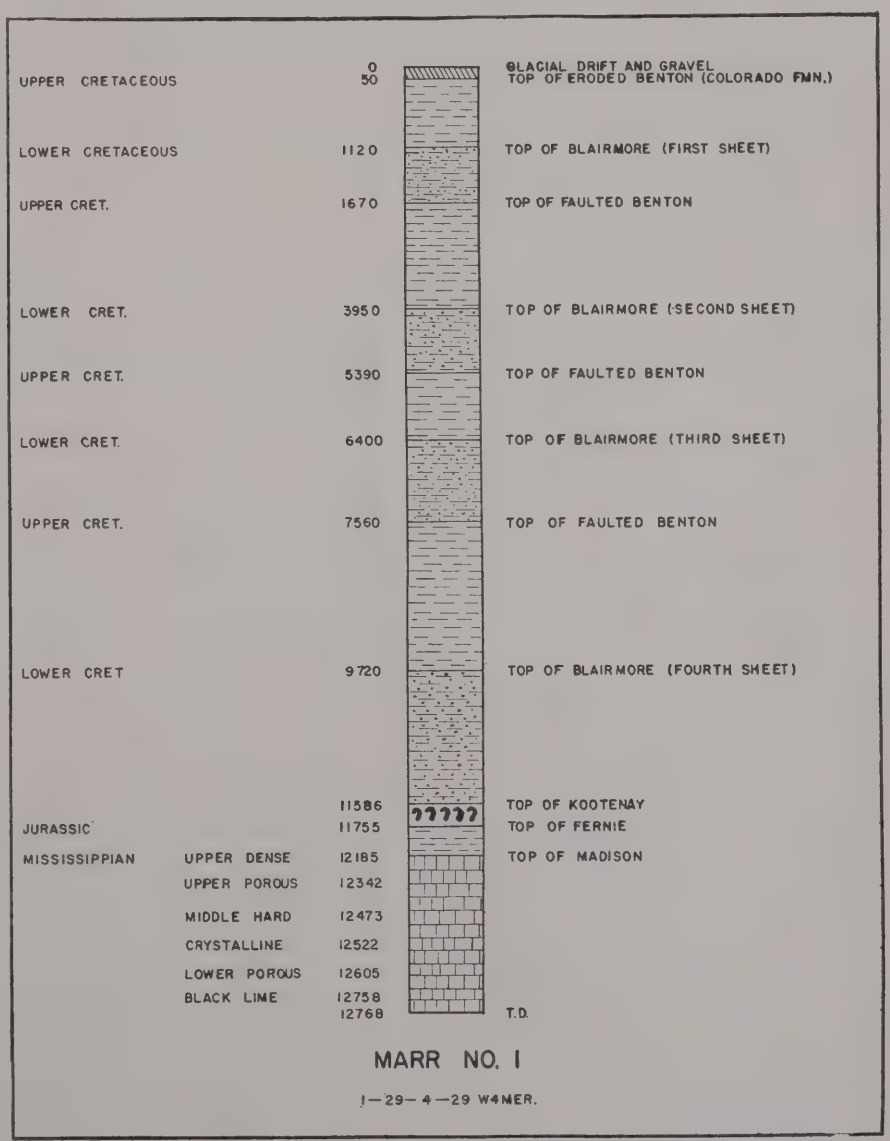

Figure 1.—Stratigraphic and structural column, Pincher Creek field.

*Canadian Gulf Oil Company, Calgary, Alta.

The Fernie formation (Jurassic) underlies the Kootenay and is composed of a brownish-black splintery shale which is slightly calcareous in part. Within this formation a 'salt-and-pepper' sandstone is encountered which contains a small amount of sweet gas. The Madison limestone of Mississippian age, encountered at 12,185 feet at the Marr well, has several members, some of which are more porous than others. The section is composed predominantly of dolomitic limestone with both fracture and intercrystalline porosity. Sulphide bands and black bituminous material are encountered throughout the section. The Madison limestone includes the Black lime which has similar physical characteristics except for little or no porosity.

Porosity Data

Table I presents a summary of the core analyses of the various lithologic members of the Madison limestone as determined by laboratory core analyses and checked by microscopic examination and evaluation. In addition to the average porosity for the section, this table gives the calculated amount of effective void space for each member. The total feet of void space for the three productive wells — Pincher Creek No. 1, W. Marr No. 1, and A. Bonertz No. 1 — is respectively 18.27 feet, 30.26 feet, and 29.12 feet, the average being 25.89 feet.

A comparison of the several lithologic members indicates that, with the exception of the middle hard and crystalline zones, comparable zones in all three wells are of approximately the same overall thickness and contain the same order of magnitude of net producing void space.

Since these three wells are separated by two to three miles and the thickness and porosity of the various lithologic members are comparable, it is felt that the average of the physical characteristics of the Madison limestone as encountered is reasonably accurate in making overall studies of the Pincher Creek field. The core from this reservoir is fractured in varying degrees throughout the entire section. However, an analysis of the porosity data indicates that approximately 90 per cent of the total void space is attributable to the intergranular system while only 10 per cent is due to the fracture system.

Table I.— Pincher Creek Gas Condensate Field
Summary of Porosity Determinations

	Pincher Creek No. 1	Marr No. 1	A. Bonertz No. 1	Average Void Space
Upper Dense—				
Interval	11,705 to 11.855 ft.	12,185 to 12,342 ft.	11,630 to 11,775 ft.	
Ft. section	150 ft.	157 ft.	145 ft.	
Ft. recovered and an lysed	54 ft.	136 ft.	137 ft.	
Plug analysis (voids)	3.18 ft.	3.81 ft.		
Whole core (voids)			5.16 ft.	
Porosity	5.89%	2.80%	3.56%	
Upper Porous—				
Interval	11,855 to 11,962 ft.	12,342 to 12,473 ft.	11,775 to 11,873 ft.	
Ft. section	107 ft.	131 ft.	98 ft.	
Ft. recovered and analysed	107 ft.	120 ft.	84 ft.	
Plug analysis	8.17 ft.	9.92 ft.		
Whole core			8.01 ft.	
Porosity	7.64%	7.57%	8.17%	
Middle Hard—				
Interval	11,962 to 12,020 ft.	12,473 to 12,522 ft.	11,873 to 11,915 ft.	
Ft. section	58 ft.	49 ft.	42 ft.	
Ft. recovered and analysed	58 ft.	33 ft.	38 ft.	
Plug analysis	2.11 ft.			
Whole core		2.06 ft.	2.04 ft.	
Porosity	3.64%	4.21%	4.85%	
Crystalline—				
Interval	12,020 to 12,100 ft.	12,522 to 12,605 ft.	11,915 to 12,009 ft.	
Ft. section	80 ft.	83 ft.	94 ft.	
Ft. recovered and analysed	59 ft.	83 ft.	66 ft.	
Plug analysis	1.76 ft.			
Whole core		7.16 ft.	8.11 ft.	
Porosity	2.20%	8.63%	8.63%	
Lower Porous—				
Interval	12,100 to 12,213 ft.	12,605 to 12,758 ft.	12,009 to 12,114 ft.	
Ft. section	113 ft.	153 ft.	105 ft.	
Ft. recovered and analysed	99 ft.	133 ft.	105 ft.	
Plug analysis	3.05 ft.			
Whole core		7.31 ft.	5.80 ft.*	
Porosity	2.70%	4.78%	5.52% *	
Total Void Space—				
Upper Dense	3.18%	3.81%	5.16%	4.05%
Upper Porous	8.17%	9.92%	8.01%	8.70%
Middle Hard	2.11%	2.06%	2.04%	2.07%
Crystalline	1.76%	7.16%	8.11%	5.68%
Lower Porous	3.05%	7.31%	5.80%	5.39%
	18.27%	30.26%	29.12%	25.89%

*Composite of whole core and plug analysis.

Permeability Data

Analysis of the core obtained at Pincher Creek No. 1 showed about 400 millidarcy-feet of formation capacity. Microscopic examination of the core indicated that 25 md.-ft. may be attributed to the intergranular porous system as there were no visible fractures in those plugs. The permeability measured in other plugs was attributed to visible fractures; such plugs, however, were not numerous. In the case of Marr No. 1, the corresponding figure is 180 md.-ft. for the intergranular system. When the core from the A. Bonertz No. 1 well was analysed it indicated that the permeability was similar to, but slightly greater than, that in Marr No. 1. In the case of the W. Marr No. 1 and Pincher Creek No. 1 cores, no permeability data were obtained on the large fractures as those core samples did not remain intact during analysis.

Of the samples showing measurable intergranular permeability, individual sample measurements ranged from about 0.1 md. (the low-

er limit of accuracy of the permeameter) to about 30 md. and averaged about 1 md. The flow tests shown in Figure No. 2 were examined to see which could be considered a reasonable representative of the flow capacity of the entire formation. Test No. 1 was taken at Pincher Creek No. 1 when only part of the productive section was exposed and the well had not been given an acid treatment to clean it up. Test No. 2 was made after a liner had been cemented in Pincher Creek No. 1 and the well capacity was obviously reduced by the mechanics of completion. Test No. 3 was made in open hole in the W. Marr No. 1, but some of the formation which was water bearing was exposed and the well had not been cleaned up sufficiently before the test. Test No. 4 was made on W. Marr No. 1 after plug back, but all the gas-productive section was exposed and the well had been given a small acid treatment to clean it up. Test No. 5 was taken at the A. Bonertz No. 1 upon completion, with the entire productive section open and without acid treatment. Test No. 6 was taken at the W. Marr No. 1 during the subsequent tests being made on the tubular equipment and chemical treatment. In some of these tests, the bottom-hole pressure has been calculated, since the bottom-hole pressure gauge failed. In view of this, it is believed that the tests which were taken with actual bottom-hole pressure measurements will be more reliable than the others. Of these tests, No. 4 appeared to be representative of the true formation characteristics and worthy of use in calculating the actual permeability of the formation. Similar calculations will be made on tests Nos. 5 and 6 at some future date. In making these calculations, the following equation* was used:

$$hk = \frac{Q_{sc}\,\mu\,log_{10}\,r_e/r_w}{0.01722\triangle P\alpha}$$

where

- hk = md.-ft. of formational capacity
- α = average relative gas density at reservoir conditions
- μ = fluid viscosity, centipoises
- r_e = drainage, radius feet
- r_w = well radius
- Q_{sc} = gas producing rate (cu.ft. per day at S.C.)
- $\triangle P$ = flowing pressure draw-down, p.s.i.

*This form of the radial flow formula for gas may be derived from equation 3 given on page 682 of Muskat's *Flow of Homogeneous Fluids* (McGraw-Hill, 1937).

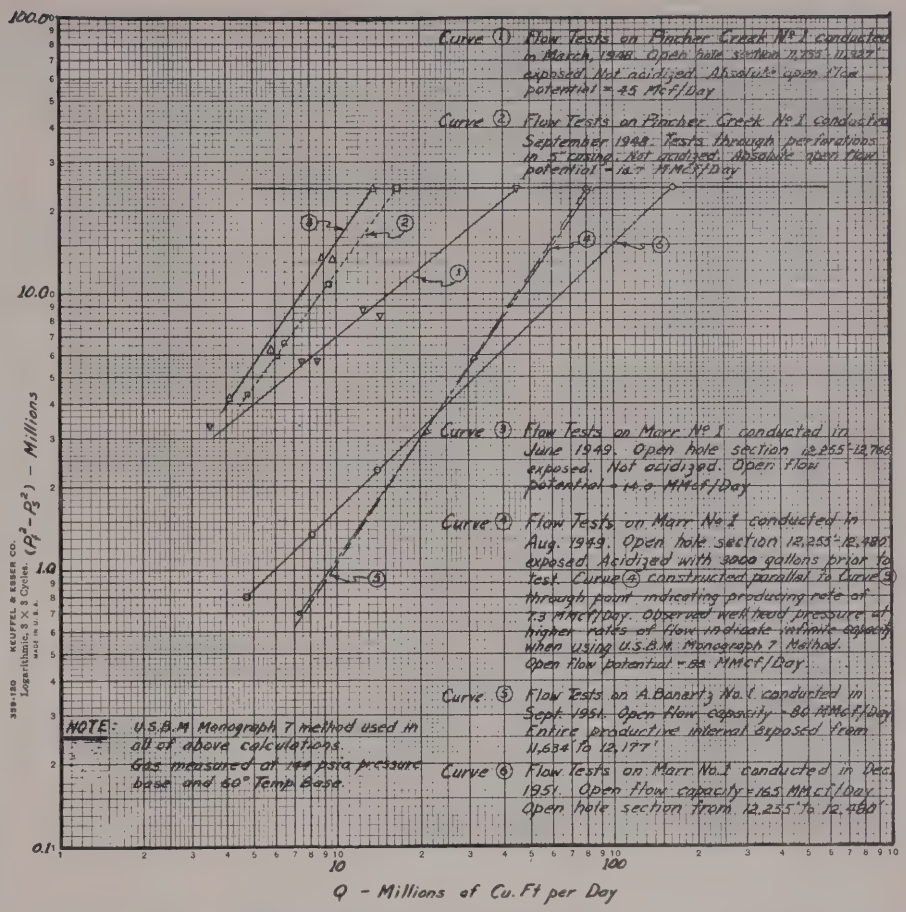
Curve ① Flow Tests on Pincher Creek No. 1 conducted in March, 1948. Open hole section 11,255'-11,427' exposed. Not acidized. Absolute open flow potential = 45 Mcf/Day
Curve ② Flow Tests on Pincher Creek No. 1 conducted September 1948. Tests through perforations in 5" casing. Not acidized. Absolute open flow potential = 14.7 MMcf/Day
Curve ③ Flow Tests on Marr No. 1 conducted in June 1949. Open hole section 12,255'-12,766' exposed. Not acidized. Open flow potential = 14.0 MMcf/Day
Curve ④ Flow Tests on Marr No. 1 conducted in Aug. 1949. Open hole section 12,255'-12,480' exposed. Acidized with 3000 gallons prior to test. Curve ④ constructed parallel to Curve ③ through point indicating producing rate of 7.3 MMcf/Day. Observed well head pressure at higher rates of flow indicate infinite capacity when using U.S.B.M. Monograph 7 Method. Open flow potential = 88 MMcf/Day.
Curve ⑤ Flow Tests on A. Bonertz No. 1 conducted in Sept. 1951. Open flow capacity = 80 MMcf/Day. Entire productive interval exposed from 11,684' to 12,177'
Curve ⑥ Flow Tests on Marr No. 1 conducted in Dec. 1951. Open flow capacity = 165 MMcf/Day. Open hole section from 12,255' to 12,480'
NOTE: U.S.B.M. Monograph 7 method used in all of above calculations. Gas measured at 14.4 psia pressure base and 60° Temp. Base.
KEUFFEL & ESSER CO. Logarithmic, 3 X 3 Cycles.
$(P_c^2 - P_s^2)$ - Millions
Q - Millions of Cu. Ft per Day

Figure 2.—Absolute open flow potentials as indicated by back pressure tests, Pincher Creek field.

Volumes and pressure drawdowns from Test No. 4 were used. The value of α was calculated, assuming the gas obeys Boyle's law as modified by the super-compressibility factor of Figure 4. The value of was estimated from published data by Bicher and Katz. The value of r_e was estimated to be 2,500 feet and r_w was estimated to be 8 feet because an acid treatment might enlarge the fractures in the vicinity of a well bore to this extent. Using this equation, the effective hk is calculated to be 1,860 md.-ft. Since the correction for relative permeability in this case is small it has been neglected.

Since the 180 md.-ft. of intergranular permeability measured on W. Marr No. 1 core samples should be reliable, it appears to be a valid conclusion that the remainder of the 1,860 md.-ft. calculated from the flow test is due to fractures. These data indicate that about 10 per cent of the formational capacity is contained in the intergranular system and about 90 per cent in the fracture system as compared with data of the section on porosity, which indicate that about 90 per cent of the effective pore volume is in the intergranular system and 10 per cent in the fractures.

As new wells are drilled and more core is obtained, a geological study of the porosity and permeability distribution will corroborate or correct these calculations.

Gas Reserves

Figure 3 is a structure contour map on the top of the Madison limestone. This map has been prepared from seismic information as corrected on the basis of the four wells that have been drilled on the Pincher Creek structure. Obviously, a structural map, which necessarily relies so heavily on seismic information — particularly at the depth at which the Madison limestone is encountered and in an area having such a complex geological history — is always subject to some question. However, it is felt that the map represents a reasonable interpretation of the best available seismic information and subsurface data. The northwestern and southeastern limits of the pool are not precisely known. On the southeast, the pool is limited by a sharp syncline, or possible fault, which has been adopted as the southeastern boundary of the

pool. On the northwest, the limit of the pool is drawn at a sharp constriction in the structure, which is assumed to be the northwestern limit of production. The essential accuracy of the map is shown by the fact that the four wells which have been drilled within the pool have encountered the top of the Madison limestone within one per cent of the estimated elevation. No radical revision of the map has been required by the results of further drilling to date. It is therefore believed that the present map is not seriously in error. As indicated on the map, the pool is limited on the east by a fault, and the estimated limit on the west is the −8,100-foot contour line. The area contained within these limits is approximately 17,250 acres. The −8,100-foot rather than the −8,200-foot contour line has been considered as the west boundary because the uppermost 100 feet of the Upper Dense zone in the F. Schrempp No. 1 well was found to have very low permeability and, for all practical purposes, this section may be considered unproductive on the west flank of the structure.

The gas-water contact in the Pincher Creek field has been reasonably well established at −8,200 feet subsea. Two wells have been drilled below this contact. F. Schrempp No. 1 was plugged back to −8,217 feet. Water was produced in all subsequent tests along with oil. This water production proves that water was present above −8,217 feet, or was so close to that level that water moved up through fissures into the well during the testing. Marr No. 1 was plugged back to −8,130 feet and no water was produced after this operation. Water was encountered in the lower part of the well and was shut off by the plug-back. It is clear that the gas-water contact lies at approximately −8,200 feet.

In calculating the overall net producing void space within the Pincher Creek structure, the average thickness and porosity of each lithologic member in the Madison limestone as encountered in the Pincher Creek No. 1, W. Marr No. 1, and A. Bonertz No. 1 wells have been used in calculating the overall gross void space of these zones where they exist above the water table.

Taking into account the structure of the reservoir in the overall area in which the various lithologic members occur above the gas-water contact, the total gross void space in

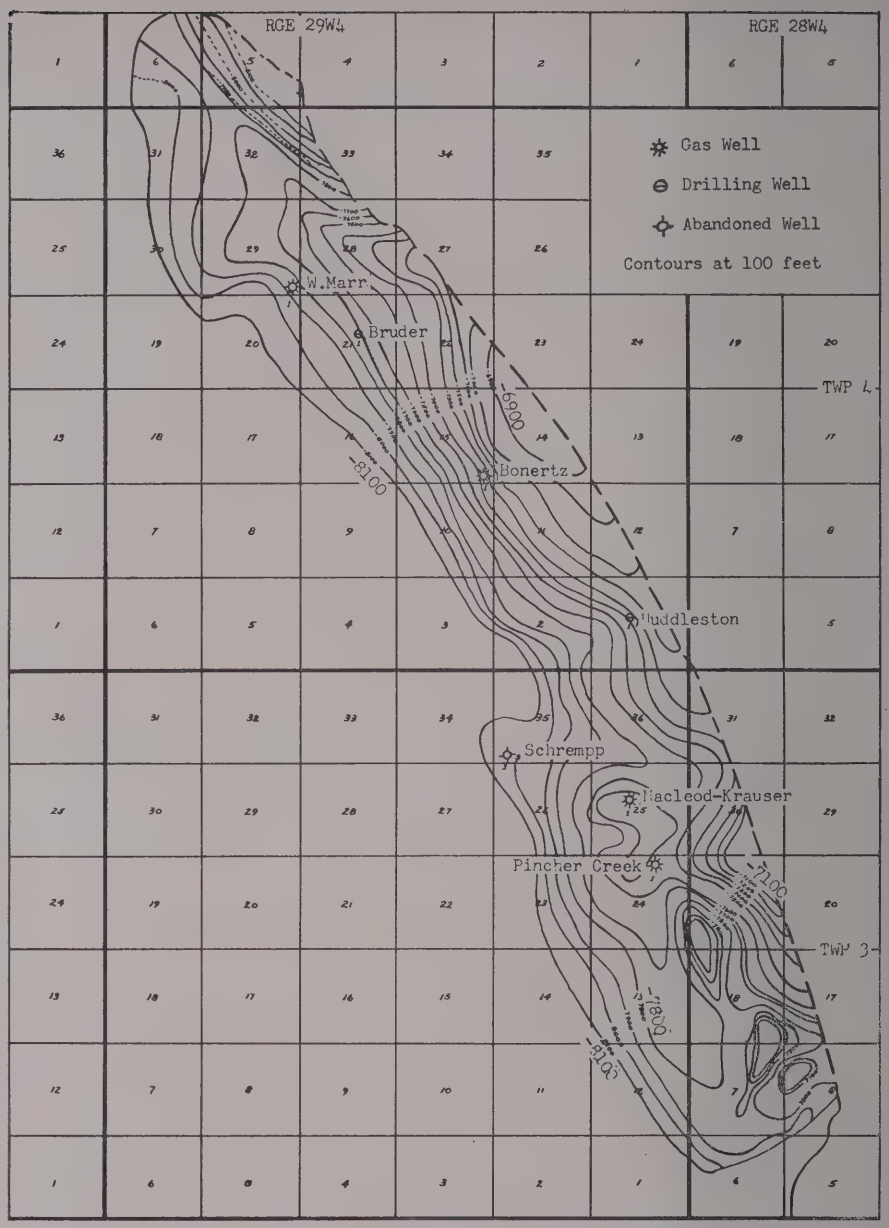

Figure 3.—Structural contour map, Pincher Creek field.
Contours on top of Madison limestone. Based on seismic information and well control

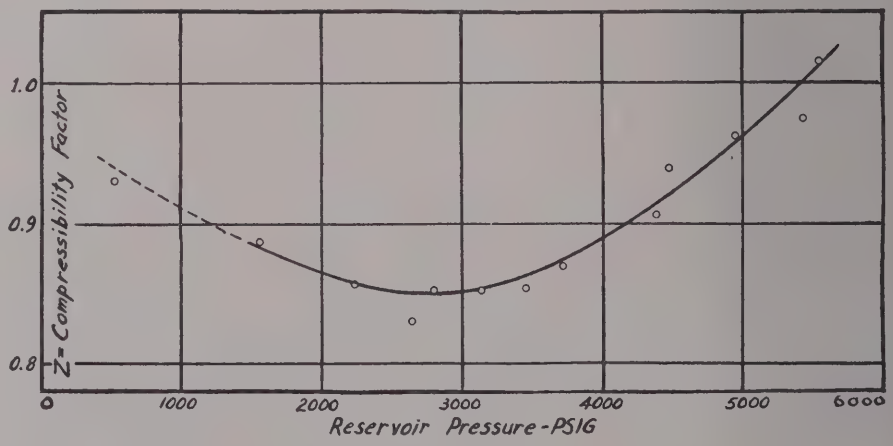

Figure 4.—Compressibility factor curve of gaseous phase at 191°F., Pincher Creek field.

the Pincher Creek reservoir is calculated to be 316,500 acre-ft. After deducting for connate water, the calculated total net effective void space for the reservoir is 218,035 acre-ft. Using the above calculation of void space, the recoverable gas in place in the Pincher Creek reservoir down

TABLE II.— SUMMARY OF PRODUCTION TEST DATA IN PINCHER CREEK FIELD

Test No.	Well	Date of Test	Length of Test Hours	Tubing Pressure P.s.i.g.	Bottom Hole Pr. P.s.i.g.	Bottom Hole Temp. °F.	Gas Prod. Rate M.c.f. /day	Cond. Prod. Rate Bbl /day	Water Prod. Rate Bbl /day	Gas-Oil Ratio Cu.ft. /bbl	Remarks
	Pincher Creek No. 1	3-26-48	—	—	4,855	184	0	0	0	0	B.h.p. gauge at 11,778 ft.
	" " "	3-26-48	6.50	1,725	—	—	12,039	363	0	33,150	Cond. blowing out thief hatch.
1	" " "	3-27-48	9.50	3,700	—	—	7,321	300	0	24,400	
	" " "	3-27-48	2.00	3,125	—	—	3,334	113.5	0	29,350	
	" " "	3-27-48	2.75	1,350	—	—	13,926	505	0	27,600	
	" " "	3-28-48	12.00	2,600	—	—	8,258	325	0	25,400	
	" " "	9-17-48	—	950	4,930	191	0	0	0	0	Diesel oil in tbg B.h.p. gauge at 12,220 ft.
	Pincher Creek No. 1	9-18-48	11.00	1,910	3,680	—	9,072	346	0	26,250	
	" " "	9-19-48	18.00	1,660	—	—	9,780	370	0	26,400	
2	" " "	9-19-48	5.00	3,000	4,485	—	4,734	177	0	26,764	
	" " "	9-19-48	1.25	2,775	4,305	—	5,986	—	0	—	
	" " "	9-20-48	15.00	—	4,225	—	6,365	242	0	26,258	
	Walter Marr No. 1	6-17, 18-49	16.50	1,480	—	—	6,970	254	384	27,441	
	" " "	6-18, 19-49	22.00	1,267	—	—	7,790	292	387	26,678	
	" " "	6-19, 20-49	25.00	2,400	—	—	4,850	184	158	26,300	
	" " "	6-20, 21-49	17.00	1,350	—	—	8,647	323	403	26,770	
3	" " "	6-21-49	4.00	3,236	4,868	188	0	0	0	0	B.h.p. gauge at 12,213 ft.
	" " "	6-21, 22-49	12.00	1,390	3,244	188	8,901	349	340	25 504	
	" " "	6-22-49	6.00	2,460	4,158	188	5,738	213	151	26,964	
	" " "	6-22-49	7.00	2,740	4,412	188	4,259	175	65	24,331	
	" " "	6-22, 23-49	17.00	1,390	3,240	188	9,725	360	310	27,014	
	" " "	7-28, 29-49	16.00	1,020	—	—	7,340	276	30	26,594	7-8-49 well PB from 12,768 ft. to,12,480 ft
	Walter Marr No. 1	8-3-49	8.50	900	—	—	6,015	263	44	22,870	Acidized with 3,000 gal. prior to this test
4	" " "	8-3,4-49	20.50	3,037	—		8,924	275	61	23,797	
	" " "	8-4-49	7.00	2,903	—	—	10,326	461	0	22 406	
	" " "	8-4, 5-49	15.00	3,043	—	—	7,439	299	0	24,880	
	A. Bonertz No. 1	9-4-51	3.50	1,000	4,528	174	21,007	—	—		
5	" "	9-4-51	6.00	800	4,219	174	31,926	—	—		
	" "	9-4-51	14.50	2,400	4 871	174	0		—		
	" "	9-9-51	6.00	615	4.369	174	33,040	—	—		Producing equipment not available for measurement of ratios.
	" "	9-9-51	6.00	775	4,762	174	15,460	—	—		
	" "	9-9-51	6.00	850	4,916	174	5,230	—	—		
	" "	9-9-51	24.00	3,300	4,952	174	0	—	—		
	W. Marr No. 1	12-18-51	6.75	3,060	4,785	190	7,026	—	—		Test conducted to determine freeze-up temperatures.
6	" "	12-18-51	6.75	2,850	4,670	190	13,800	—	—		
	" "	12-18-51	3.00	3,175	4,830	190	4,710	—	—		
	" "	12-18-51	2.00	3,350	4,900	190	0	—	—		

to a depletion pressure of 100 p.s.i.a. amounts to 2.54 trillion cu. ft. Assuming a depletion pressure of 400 p.s.i.a., the recoverable reserve is 2.40 trillion cu. ft. In each case, the figures are for a pressure base of 14.4 p.s.i.a. and a temperature base of 60°F. Of the total, 1.92 trillion cu. ft. will be net gas after the liquids and acid gases have been removed.

Well Test Data

All the production test data taken on Pincher Creek No. 1, W. Marr No. 1, and A. Bonertz No. 1 are summarized in Table II. For each of the tests made on these wells the open flow capacity was calculated by the method described in the U. S. Bureau of Mines *Monograph* 7. A plot of the calculated open flow capacity is shown in Figure 2. This figure also shows the conditions under which each test was made. It will be noted that Pincher Creek No. 1 showed an open flow capacity of 45 MM.c.f. per day before the liner was cemented. W. Marr No. 1 showed a large increase in open flow capacity from 14 MM.c.f. per day to 83 MM.c.f. per day after acid. Further tests on the Marr well have indicated the open flow capacity to be approximately 165 MM.c.f. per day based on actual bottom-hole pressure measurements. The A. Bonertz No. 1 well, completed in 1951, was tested according to the above-noted method and established an open flow capacity of 80 MM.c.f. per day without acid. These facts indicate that, with some additional perforations and acid treatment, the open-flow capacity of Pincher Creek No. 1 could be increased to a figure comparable to those of W. Marr No. 1 and A. Bonertz No. 1. It seems reasonable to expect that more wells will be completed with capacities in the order of magnitude of 80 MM.c.f. per day capacity.

Reservoir Fluid Characteristics

At the time of the completion of Pincher Creek No. 1, several duplicate samples of separator liquid and gas were obtained. These samples, contained in standard steel containers, were shipped to the Gulf Research and Development Company for analysis. Because of the corrosive nature of the fluids there has been some difficulty in obtaining duplicate fractional analyses on the various samples. Considering this difficulty, it is felt that the

TABLE III.— FRACTIONAL ANALYSIS OF PINCHER CREEK FLUID

Component	Reservoir Fluid Mol. %	Separator Gas Mol. %	Gas After Treating Mol. %	Dry Gas Mol. %
Methane	74.20	77.21	92.80	95.40
Ethane	3.27	3.29	3.95	3.42
Propane	1.21	1.12	1.35	0.64
Butane	0.83	0.65	0.78	0.00
Pentane	0.57	0.33	0.40	0.00
Hexane	0.45	0.13	0.16	0.00
Heptane Plus	2.14	0.05	0.06	0.00
Carbon Dioxide	6.16	6.30	0.00	0.00
Hydrogen Sulphide	10.77	10.50	0.00	0.00
Nitrogen	0.40	0.42	0.50	0.54
Total	100.00	100.00	100.00	100.00

original analysis which was made on a relatively fresh sample is probably most representative of the reservoir fluid. Table III presents the fractional analyses so obtained. All of the analyses on Pincher Creek fluid are rather unusual in that the *C7*-plus fraction is relatively large considering the very high concentration of methane in the hydrocarbon system. The concentration of propane and butane is relatively low. This situation indicates that the recoveries to be anticipated by processing the separator gas in an absorption plant would be correspondingly low, whereas the stock tank recovery condensate would be relatively high. In addition to the routine fractional analyses, an exhaustive series of tests was conducted to determine the existent nature of the sulphide compounds in the fluid. These tests prove conclusively that there is less than 1 grain mercaptans and other organic sulphide compounds per 100 cu. ft. of gas. Thus, for all practical purposes, the sulphide content of Pincher Creek fluid exists entirely as hydrogen sulphide.

Figure 4 shows the variation in 'compressibility factor' of the gaseous phase of the reservoir fluid as a function of pressure at isothermal conditions of 191°F. Obviously, the curve so depicted is not a true compressibility factor, since the composition of the gaseous phase is a function of pressure. However, the Z factor displayed is applicable in the calculation of the volume of gas phase in place at any reservoir pressure.

The fact that the Pincher Creek reservoir fluid is of the retrograde condensate type is shown in Figure 5. This plot reflects the change in well effluent composition by isothermal pressure depletion. In the laboratory investigation from which the basic data for the plot on Figure 5 were determined, multi-stage separation was outlined, with a primary separator operating at 750 p.s.i. and 60°F.; intermediate stage separator operating at 50 p.s.i. and 60°F.; and stock tank operating at 60°F. with atmospheric pressure. In order to obtain this basic information on the Pincher Creek fluid it was necessary to re-design the laboratory equipment, and to date a complete investigation on only a single sample has been made. Other samples are now being analysed. Consequently, until duplicate investigations have been carried out, the phase behaviour as shown in Figure 5 must be considered as tentative.

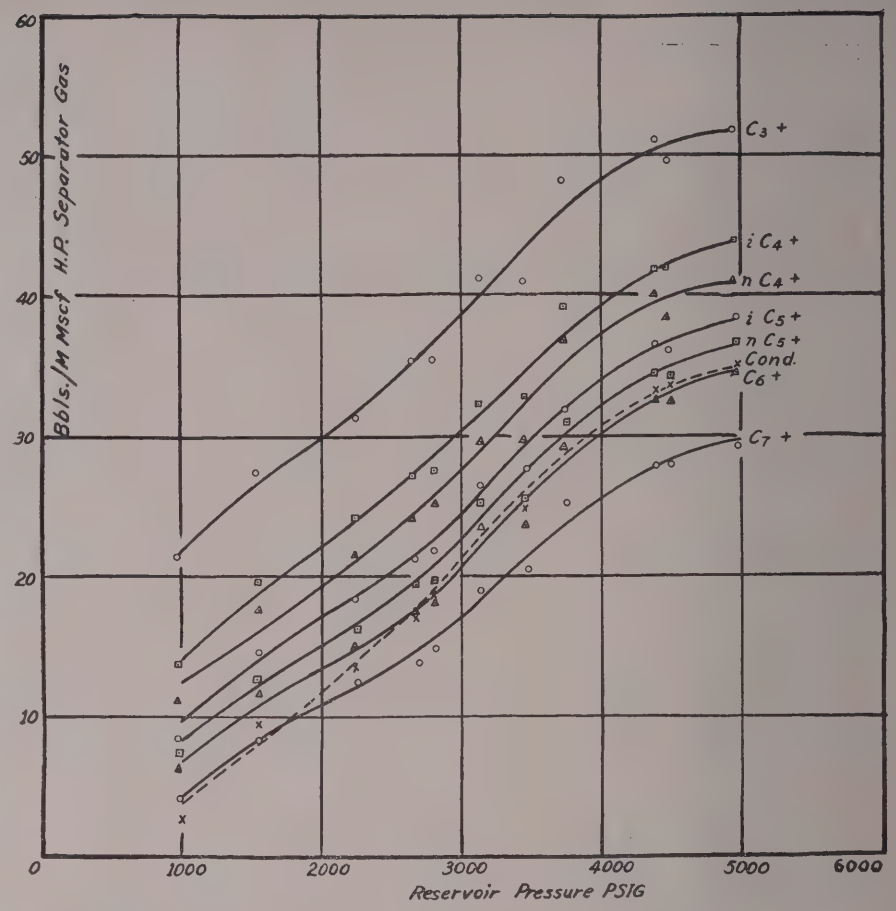

Figure 5.—Well effluent composition by isothermal pressure depletion, temperature 191°F., Pincher Creek field.

Calculated Reservoir Performance

After having calculated the reserves of this pool and determined the fluid characteristics by model reservoir studies, it has been possible to calculate the reservoir performance by pressure depletion and cycling methods. Only calculations of the pressure depletion method are discussed here. Figure 6 illustrates this type of depletion, with the history being carried for 44 years to a pressure point of 400 p.s.i.a. Figure 6 also illustrates the decline in reservoir pressure, decline in condensate content of the gas, and cumulative condensate recovery for a producing rate of 200 MM.c.f. per day. It was estimated that, in order to maintain a rate of 200 MM.c.f. per day for a reasonable length of time under pressure depletion operation, it would be necessary to complete a minimum of ten wells and equip them with 4-inch o.d. tubing. In turn, these wells would have to be produced at 20 MM.c.f. per day per well, which is approximately 25 per cent of the average open flow capacity. Additional wells would necessarily be drilled to maintain this production rate and to provide standby wells during workovers which would occur during the life of the pool. Perhaps twenty to twenty-four wells will ultimately be required to effectively drain the reservoir. The reservoir pressure was calculated at 2-year intervals, assuming that the gas obeys Boyle's law modified by the super-compressibility factors shown in Figure 4. With the pressure history known, the history of the condensate content of the gas produced may be determined by application of Figure 5. From these data, the rate of condensate production and cumulative condensate recovery may be calculated. Since the acid gases constitute about 16 per cent of the total gas volume, and miscellaneous fuel requirements are estimated at 4 per cent, the remaining 80 per cent was considered salable. On the basis of 2.40 trillion cu. ft. of reservoir gas, 1.92 trillion cu ft. of sweetened dry gas would be available for sale. The other 20 per cent of this total would contribute to the following amounts of solids and liquids:

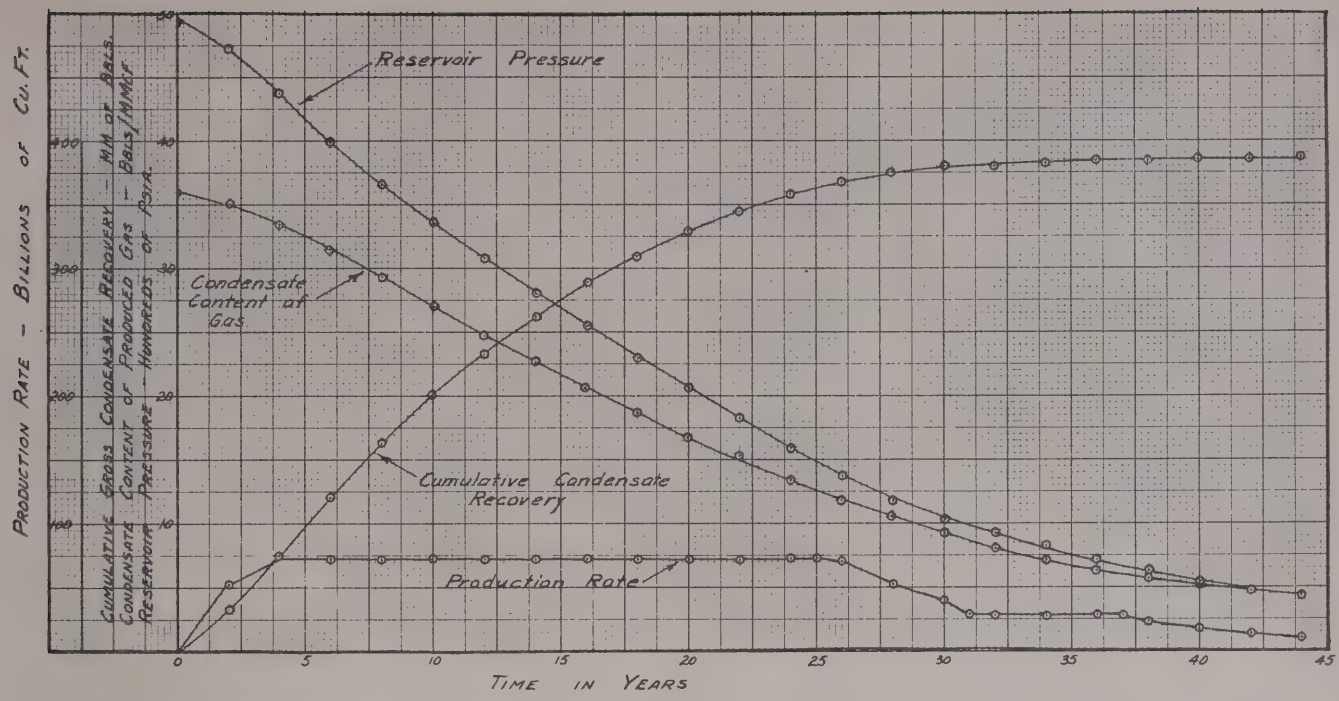
Production Rate - Billions of Cu. Ft.
Cumulative Gross Condensate Recovery - Mm. of Bbls.
Condensate Content of Produced Gas - Bbls./MMcf
Reservoir Pressure - Hundreds of Psia.
Reservoir Pressure
Condensate Content of Gas
Cumulative Condensate Recovery
Production Rate
Time in Years

Figure 6.—Estimated producing history, Pincher Creek reservoir. Pressure depletion at a rate of 200 million cu. ft./day.

Condensate oil....38.4 million barrels
Natural gasoline.. 9.8 million barrels
Propane.....12.7 million barrels
Butane.....10.4 million barrels
Sulphur..... 7.4 million long tons

These components are, of course, salable in their produced form or in products which may be developed by further refining and through processing as petrochemicals.

Petrochemical Possibilities of Pincher Creek

The marketing of the various by-products listed above may form the basis for an important petrochemical industry in Alberta. It is assumed that condensate, oil, and natural gasoline will be mingled with refinery products to produce motor fuel and Diesel oil for use in Alberta. Preliminary surveys indicate that all the propane will be needed for cooking, water heating, household fuel, and tractor operation in Alberta. Under present conditions, it would be very difficult to dispose of the great volume of sulphur which will be produced. Alberta and British Columbia together consume only 50,000 tons annually, or 1,500,000 tons at the present rate during the thirty years' prospective life of the Pincher Creek pool. The principal market for sulphur in Canada is in Ontario and Quebec. Sulphur is now delivered in Ontario for $30.00 per ton from the Texas coast. It appears that it will be impossible to ship the sulphur eastward to Ontario because of the freight charges involved in such movement. A market may be developed in the northwestern part of the United States, but such a market must compete with the sulphur produced by gas wells, refineries, and smelters in Wyoming and Montana. Fortunately, sulphur can be stored in stockpiles on the surface, without deterioration and at very low cost. It may be necessary to store much of the sulphur produced at Pincher Creek for lack of an adequate market.

The butane production and the ethane content of the gas are the most probable bases for a petrochemical industry. The raw gas contains approximately 3.3 per cent ethane, so that a production of 6,600,000 cubic feet daily is theoretically possible. The extraction of ethane is difficult and expensive. It is doubtful whether more than 40 per cent of the ethane can be economically recovered under present conditions. Accordingly, the maximum volume of available ethane appears to be about 2,640,000 cubic feet daily.

In an effort to learn more about the petrochemical possibilities of Pincher Creek, the writer recently visited the extensive petrochemical industry in south Texas. As is well known, this industry is based primarily on gases resulting from cracking operations in refineries. Secondarily, the petrochemical industry uses liquefied petroleum gases, such as propane and butane obtained from natural gasoline plants and from refinery operations. Methane is used primarily for fuel consumption and for the production of relatively small amounts of ammonia and methanol. These are products of low value per pound and, accordingly, are not suitable for manufacture at Pincher Creek. Freight charges on either ammonia or methanol from Pincher Creek to consuming centres would exceed the cost of producing these materials in the industrial area of eastern Canada. Currently, the Consolidated Mining and Smelting Company is producing a large volume of ammonia at Trail, British Columbia This ammonia is needed to combine with the sulphur output of the Trail smelter. It is understood that less than 10 per cent of the resulting ammonium sulphate finds a market in Canada. The balance is exported to Washington, Oregon, and California. There is little prospect for developing much additional ammonia demand in Alberta. The high cost of moving ammonia to Vancouver and then delivering it to West Coast consuming areas in the United States makes it impractical to produce ammonia at Pincher Creek. The low-cost production in the western United States would eliminate any possible competition from Pincher Creek.

Table IV lists the more important petrochemical plants in Texas. The

predominance of plants using refinery gases is evident from the table. The absence of important plants using methane illustrates the previous statement that these low-value products are most economically manufactured close to the point of consumption rather than near the gas producing fields. Even with the advantage of low-cost water transportation from the ports along the Gulf Coast, the petrochemical industry has not found it advantageous to build plants using methane in important amounts in Texas.

At the present time, it appears most likely that Pincher Creek could support a petrochemical industry based on ethylene, because of the relatively high value per pound of the end products. Ethylene can be made from ethane by dehydrogenation. Similarly, ethylene is made by cracking butane. Currently, there is only one known plant in Texas operating on ethane. One plant is under construction which is designed to crack ethane, propane, and butane. The market for the resulting ethylene is excellent, and it is understood that one producer is considering plans for building an additional plant to use ethane, propane, and butane. There is a rapidly expanding demand for polyethylene, produced from ethylene. This product sells for approximately 50 cents per pound. It can support a relatively high freight charge and still be delivered at consuming centres in competition with existing sources of supply.

The construction of an ethylene plant obviously depends upon the assurance of an adequate, steady, and long-lived supply. If Pincher Creek produces 200,000,000 feet of raw gas daily, the ethane and butane available could produce 240,000 pounds per day of ethylene. This is approximately 10 per cent of the ethylene produced in Texas from all sources. The 240,000 pounds of possible daily ethylene production corresponds to 100 per cent of the ethylene glycol which is used for permanent-type anti-freeze now consumed in Canada, plus 100 per cent of the ethyl alcohol (grain alcohol) produced in this country. Pincher Creek accordingly offers the possibility of producing Canada's entire needs for ethylene glycol and ethyl alcohol. A plant at Pincher Creek producing 240,000 pounds per day of ethylene and converting to ethylene glycol and ethyl alcohol is estimated to cost $12,000,000. The additional dry gas required for fuel and energy would be only 12,000,000 feet per day.

TABLE IV.— TEXAS' LARGER PETROCHEMICAL PLANTS

COMPANY	LOCATION	PRINCIPAL PRODUCTS
PLANTS USING GASES AND PETROLEUM AS RAW MATERIALS		
Allied Chemical and Dye Co.	Orange	Ethylene glycol, ethylene oxide
Atlanta Refining Co.	Atreco	Detergents
*Carbide and Carbon Co.	Texas City	Ethylene glycol, polyethylene, styrene, methanol
Cosden Petroleum Corp.	Big Spring	Mercaptans
Diamond Alkali Co.	Deer Park	Chlorinated ethylene products
Ethyl-Dow Co.	Freeport	Ethylene dibromide, styrene
E. I. Dupont de Nemours Co.	Houston	Fungicides, insecticides
E. I. Dupont de Nemours Co.	La Porte	Benzene derivatives, agricultural chemicals
Eastern States Petroleum Co.	Houston	Benzene
Ethyl Corporation	Houston	Tetraethyl lead, ethyl chloride
General Tire and Rubber Co.	Baytown	Synthetic rubber
Goodrich Chemical Co.	Port Neches	Synthetic rubber
Goodyear Synthetic Rubber Co.	Houston	Synthetic rubber
Great Southern Chemical Co.	Corpus Christi	Benzene, toluene
Gulf Oil Corporation	Port Arthur	Iso-octyl alcohol, ethylene
Humble Oil Refining Company	Baytown	Butyl rubber, solvents, xylenes.
Jefferson Chemical Co.	Port Neches	Ethylene products, alkylated phenols
Koppers Company	Port Arthur	Ethyl benzene for synthetic rubber
Magnolia Petroleum Corp.	Beaumont	Ethylene, solvents
*Monsanto Chemical Co.	Texas City	Styrene, synthetic fibres
Neches Butane Products Co.	Port Neches	Butadiene
Pan American Chemical Div.	Texas City	Benzene, toluene, varnish substitutes
Pure Oil Company	Nederland	Butenes
Republic Oil and Refining Co.	Texas City	Benzene, toluene, solvents
Shell Chemical Co.	Houston	Acetone, alcohols, glycerine
*Shell Oil Co.	Houston	Benzene, toluene
Sinclair Refining Co.	Houston	Butadiene
Sunray Oil Corp.	Corpus Christi	Detergents
The Texas Company	Port Arthur	Naphthenic acid.
United States Rubber Co.	Port Neches	Synthetic rubber
*Also uses propane and/or butane		
PLANTS USING PROPANE AND BUTANE PRIMARILY		
Celanese Corporation	Bishop	Propanol, butanol and aldehydes
Celanese Corporation	Pampa	Solvents and aldehydes
Dow Chemical Company	Freeport	Ehylene glycol, plastics, styrene.
*E. I. Dupont de Nemours	Orange	Polythene, nylon salts
*E. I. Dupont de Nemours	Victoria	Nylon base chemicals
Monsanto Chemical Co.	Texas City	Styrene, acrylonitrile, vinyl chl.
Phillips Petroleum Co.	Borger	Butadiene, nylon base chemicals
Phillips Petroleum Co.	Borger	Synthetic rubber
Texas Eastman Corp.	Longview	Aldehydes, ethanol, plastics
*Also uses refinery bases or derivatives.		
PLANTS USING NATURAL GAS PRIMARILY		
Carthage Hydrocol, Inc.	Brownsville	Gasoline and Fischer-Tropsch chemicals
Diamond Alkali Co.	Deer Park	Ammonia
E. I. Dupont de Nemour	Orange	Methanol
Hercules Powder Company	Houston	Ammonia
Heyden Chemical Company	Houston	Methanol
Mathieson Chemical Co.	Houston	Ammonia
McCarthy Chemical Co.	Winnie	Methanol, aldehydes
Phillips Petroleum Co.	Etter	Ammonia
Rohm and Haas	Houston	Cyanides
San Jacinto Chemical Corp.	Houston	Ammonia

It should be noted that two of the larger plants using natural gas have recently been shut down for process changes. They may not be commercially successful plants.

SUMMARY

	NUMBER	ESTIMATED COST
Plants using refinery gases and petroleum	30	$666,260,000
Plants using propane and butane (LPG)	9	334,400,000
Plants using natural gas primarily	10	88,026,000
TOTAL	49	$1,088,686,000

An attempt has been made to estimate the ethylene demand in northwestern United States. Sufficient data are not available. If it is assumed that the market demand in this area is proportionate to the population, it appears that the five northwestern States would consume 70,000,000 pounds annually. This is about 80 per cent of the 87,000,000 pounds of production which has been estimated as the Pincher Creek potential. It may be speculated that Pincher Creek could export to the northwestern United States all their needs for ethylene glycol anti-freeze, ethylene oxide, and styrene for artificial rubber and similar materials if the Canadian market would not absorb the Pincher Creek production.

Of course, the ethylene cannot be produced except concurrently with the production and sale of approximately 160,000,000 feet per day of dry gas.

Acknowledgment

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Studies in Petrology, Goldfields Area, Saskatchewan*

By RALPH W. EDIE†

(*Transactions, Volume LV,* 1952, pp. 396-405)

Introduction

THE GOLDFIELDS AREA in northern Saskatchewan has become important in recent years because of numerous discoveries of pitchblende. The area is underlain by rocks of Precambrian age and, because of metamorphism and hydrothermal alteration, the geology in the vicinity of the pitchblende deposits is exceedingly complex and presents a challenge to all geologists working in the district. This paper is a contribution to the geological literature of the area and presents a description of rock types, supplemented with petrographic, chemical, and spectrochemical data.

Field work in connection with this paper was carried out during the summers of 1948, 1949, and 1950 while the writer was employed as assistant geologist for the Crown Corporation, Eldorado Mining and Refining (1944), Limited, Goldfields area. Laboratory work was undertaken at the Massachusetts Institute of Technology during the winter months of 1949-50 and 1950-51 and extended through the summer of 1951.

With the exception of SiO_2 and FeO, all of the new analyses (1) presented were carried out by quantitative spectrographic techniques under the supervision of Dr. L. H. Ahrens at M.I.T. Many of the techniques have only recently been developed at that Institute and detailed laboratory procedures have not yet been published.

*Part of a Ph.D. thesis submitted to the Massachusetts Institute of Technology, November, 1951; H. W. FAIRBAIRN, thesis supervisor.

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(1) Al_2O_3, total Fe, MnO, MgO, CaO, Na_2O, K_2O, Rb_2O, Li_2O, Cu, Co, and Ni were determined spectrographically by the writer.

F and Ga_2O_3 were determined spectrographically by R. Seraphim and C. K. Bell.

SiO_2 and FeO were determined chemically for the writer at the University of Minnesota.

General Geology

Excellent reviews of the geology of the Goldfields area have been given by Alcock (3) and by Christie and Kesten (5). Their reports formed the basis for further detailed geological work in the area.

Table I shows the probable geological succession in the areas studied.

The Tazin group comprises an assemblage of sediments, lava flows, and basic intrusive rocks that have been regionally metamorphosed and intruded by granitic rocks of batholithic extent. Several types of granitic rocks are recognized, including granodiorite, granite, and alaskite, but the relationship between these rocks is not known.

After erosion had bared large areas of granitic rock, the sediments and volcanics of the Athabaska series were laid down on the unconformity. They consist of interbedded conglomerate, arkose, and basalt flows, with a few intercalated basic sills.

Diabase dykes, which are numerous in the area, are believed to represent feeder dykes to flows and sills. It is possible that some dykes are younger than any flows or sills now exposed.

Athabaska time closed with a period of folding, with resulting rather broad, open folds whose axes trend north-northeast. This was followed by faulting along northeast trending lines. The faulting was accompanied by extensive shearing, mylonitization, and brecciation and is believed to have provided the channelways for hydrothermal alteration and uranium mineralization.

In the area north of lake Athabaska, most of the rocks of the Athabaska series have been eroded away since late Precambrian time except where local post-Athabaska folding has preserved synclines, an excellent example of which is the syncline at Martin lake (Figure 1).

(3) For references see end of paper.

TABLE I.— TABLE OF FORMATIONS

QUATERNARY		Gravel, sand, boulder clay
	Erosion	
PROTEROZOIC	Athabaska* series	Basalt flows, sills (?), and diabase dykes Arkose Conglomerate
	Unconformity	
	Intrusive rocks	Granite and granite-gneiss
	INTRUSIVE CONTACT	
PROTEROZOIC or ARCHAEAN	Tazin* group	Mafic inclusions in alaskite Plagioclase-amphibolite Garnet-muscovite schist Quartz-muscovite schist Chlorite-sericite schist Slate Quartzite

*The sequence of the various rock types in the Tazin group and Athabaska series, as given in the Table, is arbitrary.

Linear remnants of Athabaska rocks have also been preserved at several places in the area along the southeast side of northeast trending faults.

Petrology

Quartzite

Quartzite outcrops at a number of places on the Eagle-Ato-Mic claims (see Figure 2). The rock is generally massive and bedding is absent or only faintly recognizable. It is normally white to smoky grey, but locally is purple due to small amounts of disseminated hematite.

Results of an analysis of mylonitized quartzite from the Eagle mine are given in Table II. The rock analyzed is not bedded and contains quartz, chlorite, and epidote. Thin section study suggests that at least part of the chlorite and possibly all of the epidote were introduced hydrothermally into minute fractures. If this is true, the percentage of calcium, magnesium, and iron in the typical quartzite of the area may be lower than shown in the analysis.

Table II.— Analysis and Mineral Composition of Quartzite, Eagle Mine

SiO_2	89.1 per cent
TiO_2	n.d.
Al_2O_3	6.7 "
Fe_2O_3	2.1 "
FeO	1.4 "
MnO	0.1 "
MgO	2.0 "
CaO	0.7 "
Na_2O	0.5 "
K_2O	0.2 "
H_2O	n.d.
CO_2	n.d.
	102.8
F	0.007 per cent
Ga_2O_3	0.00075 "
Rb_2O	0.0026 "

Mineral Composition

Quartz	89.5 per cent
Unidentified opaque	0.4 "
Chlorite and sericite	10.0 "
Calcite	0.1 "
Epidote	Trace

On the Eagle property, rocks mapped as quartzite may originally have been orthoquartzite, chert beds, or local silicified zones. In general, these rocks contain 90 per cent or more quartz with a few shreds of sericite. Of several thin sections examined, none showed secondary enlargement of quartz grains or heavy residual minerals common to orthoquartzites. However, the rocks examined were considerably mylonitized. Such features as secondary enlargement or recognizable heavy residual minerals could have been erased by cataclastic metamorphism. Their absence, therefore, is not significant in determining the origin of the rock.

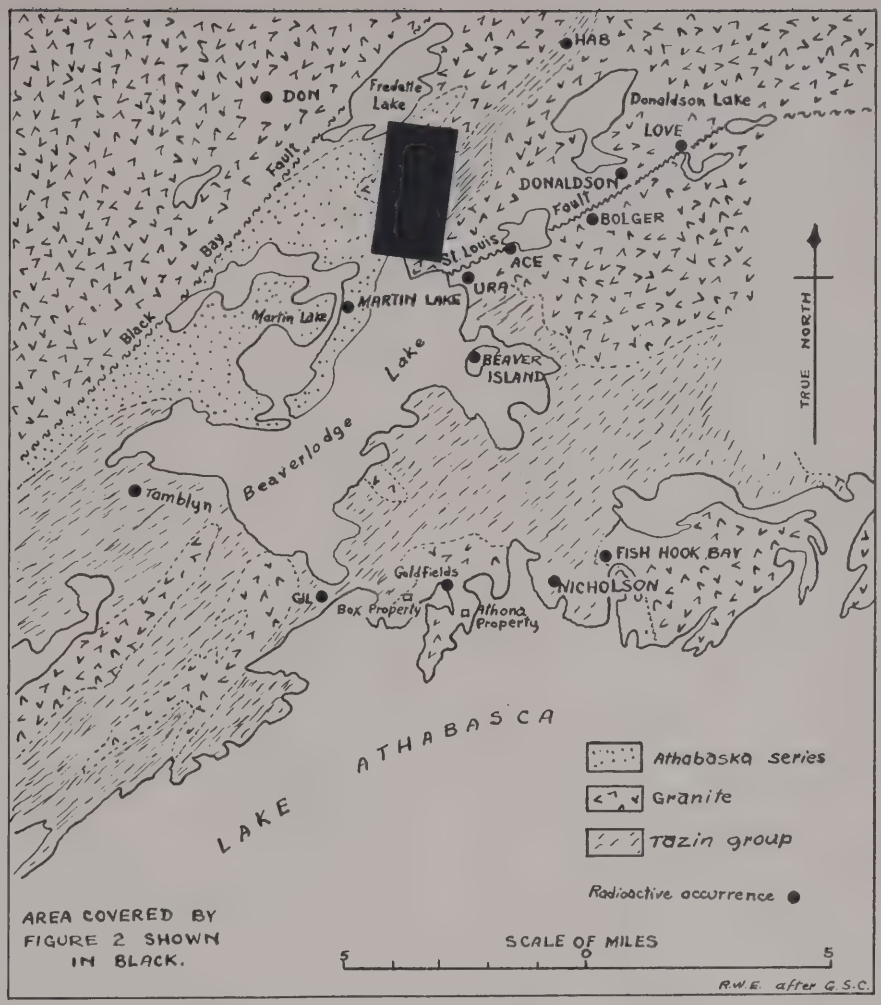


Figure 1.—Sketch-map showing general geology of the Goldfields area.

Slate

Slate underlies a large area on the Eagle-Ato-Mic claims (see Figure 2). The rock is generally dark-grey to black and at many localities bedding is not clearly visible. However, on the southern part of the claims, the slate is finely bedded and the bedding is nearly parallel with the slaty cleavage.

In thin section, the slates are seen to consist predominantly of chlorite and white mica, with lesser amounts of quartz, feldspar, and epidote.

An analysis of typical bedded slate is given in column *D* of Table III. For comparison, analyses by Pettijohn of Knife Lake slate and Glenwood shale are given in columns *F* and *E*, respectively. The analyses indicate that both the slate from the Goldfields area and the Knife Lake slate probably were originally normal shale-type sedimentary rocks.

Chlorite-Sericite Schist

Chlorite-sericite schist is found in several widely separated localities on the Eagle-Ato-Mic claims. A typical specimen is pale greenish-grey and has finely laminated bedding.

Microscopic study shows that these rocks are composed chiefly of chlorite and white mica, with minor amounts of albite, quartz, and epidote. Because of similar mineralogical composition, the chlorite-sericite schists are believed to be slightly higher grade metamorphic equivalents of the black slates described above.

At several places near the Ace mine, bands of chlorite-sericite schist occur in plagioclase amphibolite. The bands range from a few inches to several feet thick, are thinly bedded, and are whitish-grey to greenish-grey.

A large mass of chlorite-sericite schist is exposed in the underground workings at the Ace mine. This material is petrographically similar to the chlorite-sericite schist that occurs as bands in plagioclase amphibolite and, like it, is thinly bedded.

The chlorite-sericite schist in the Ace mine, and the plagioclase amphibolite nearby, have been intruded by alaskite. Tabular and lenticular masses of the alaskite, now altered to a brick-red mylonite, occur within the schist. They lie in the plane of schistosity of the St. Louis fault zone. This probably does not mean that sill-like injections of alaskite entered the schist parallel to present schistosity. On the contrary, it is likely that the original injections were of an irregular nature but were rolled out (mylonitized) into tabular sheets parallel to the newly developed schistosity which was superimposed on all rocks in the fault zone.

Microscopic study shows that the chlorite-sericite schist on the Eagle-Ato-Mic claims consists chiefly of chlorite, sericite, and albite, and minor amounts of quartz and epidote. Thus, in mineralogical composition, it is similar to the slates and argillites on these claims.

In the Ace mine, the composition of the large mass of chlorite-sericite schist is not uniform. Within the schist are bands and streaks of brown to red cherty-looking rock. Under the microscope, the brown rock appears to be a mechanical mixture of broken veinlets of alaskite in swirls of chlorite-sericite schist. The brown cherty bands were probably *lit-par-lit* injection gneisses within the main mass of chlorite-sericite schist before the St. Louis fault zone was developed.

A comparison of an analysis of the brown rock (Table III, Col. *C*) with that of chlorite-sericite schist (col. *B*) shows an increase in silica in the former rock with a concomitant decrease in all other constituents except Na_2O and CaO. These chemical differences are probably in large part due to 'granitization', but it must be kept in mind that at least some differences are probably due to later hydrothermal alteration.

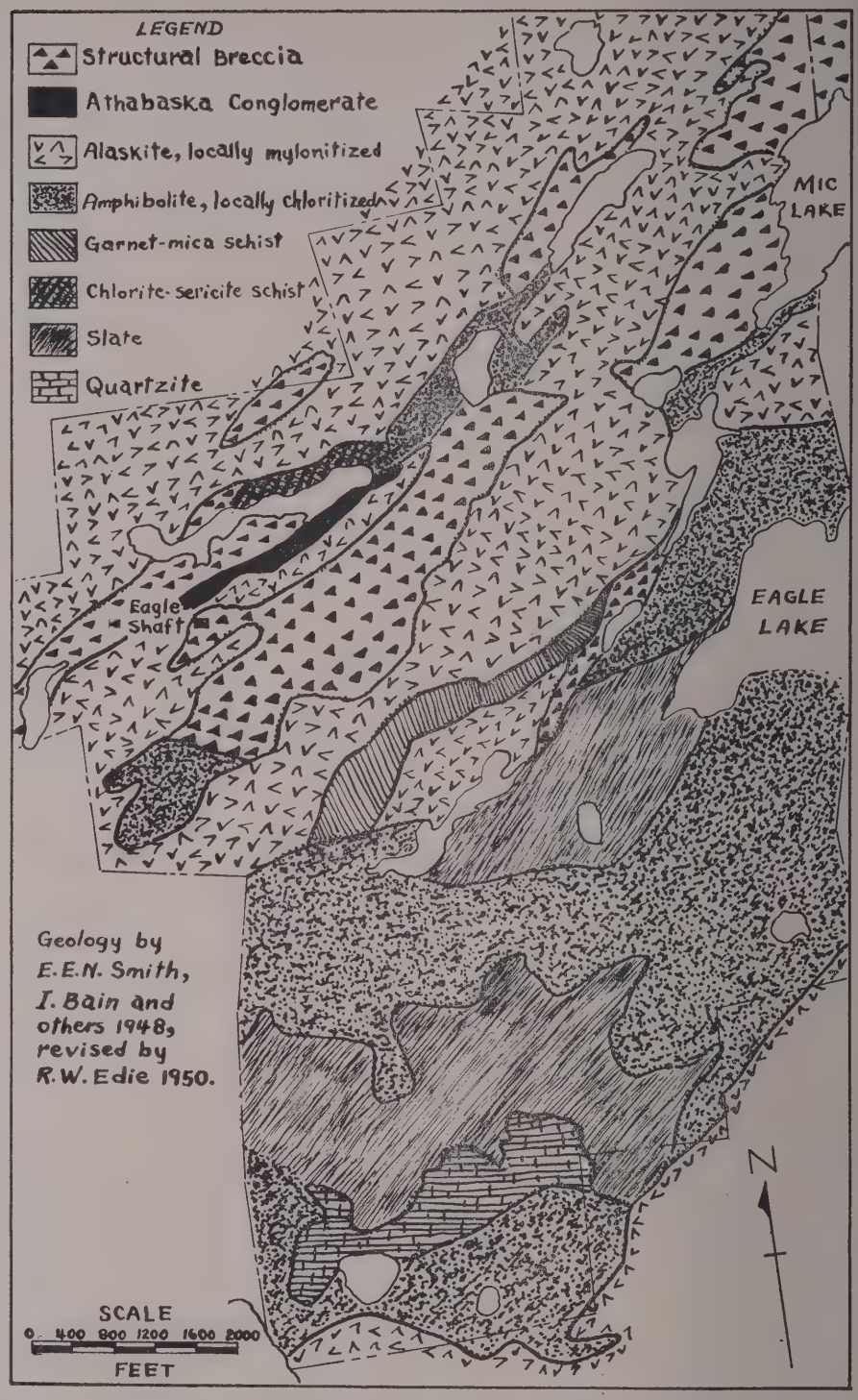


Figure 2.—Sketch-map showing geology, Eagle-Ato-Mic claims.

Quartz-Muscovite Schist

A specimen of quartz-muscovite schist from the Tamblyn property (see Figure 1) was analyzed for the purpose of getting an estimate of the effect of hydrothermal alteration in a nearby structural breccia of the same material. The rock is honey-yellow, medium to coarse grained, and highly schistose, with bedding only faintly discernable.

An analysis and the calculated mineralogical composition of the rock are shown in Table III, col. *A*). The analysis indicates that the mica in the schist is a true muscovite, as potash is high compared to soda.

Garnet-Muscovite Schist

A belt of garnet-muscovite schist, several hundred feet wide, occurs on the Eagle-Ato-Mic claims, near the centre of the claim group (Figure 2). The rock typically has a honey-yellow colour. Dark-red garnets up to 1 mm. in diameter are common but are not easily recognized without a hand lens.

In thin section, this rock is seen to be composed largely of muscovite and quartz, with minor amounts of garnet. Smith (12) found considerable cordierite in rocks of this type, but none was observed in thin sections examined by the writer.

The mineralogical composition indicates that this rock was probably a shale-type of sedimentary rock and that it may be a higher grade metamorphic equivalent of the slates and chlorite-sericite schists of the area.

Plagioclase Amphibolite

Plagioclase amphibolite is widely distributed on the Eagle-Ato-Mic claims (Figure 2). In a few localities, narrow bands of metamorphosed sediments are found within

the rock. The sediments (bedded chlorite-sericite schists) are lighter in colour than the adjacent plagioclase amphibolite and can rarely be traced for more than 20 to 30 feet along the strike.

On the Eagle and Ace properties, no field evidence was observed indicating the original nature of the plagioclase amphibolites before metamorphism. They could be of either intrusive or extrusive origin. It may be noted, however, that rocks of similar type, showing well-defined pillow structure, are described by Alcock (3) from the southern part of the Goldfields area, and massive, coarse grained plagioclase amphibolites from this same region have been designated 'metagabbro' by Christie (4).

In hand specimen, the coarser grained plagioclase amphibolites have a speckled appearance due to the intermingling of grains of plagioclase (white) and hornblende (greenish-black). Finer grained specimens have a uniformly dark-grey to yellowish-green colour.

A study of thin sections shows that the plagioclase amphibolites have been completely recrystallized. They are composed chiefly of blue-green hornblende and untwinned andesine or oligoclase, with minor amounts of chlorite, epidote, and opaque minerals.

The grain size depends on the degree of crushing to which these rocks were subjected during late cataclastic metamorphism. Where crushing has been severe, hornblende crystals are splintered and broken and the texture has been transformed to a felted mass of hornblende fragments with interstitial plagioclase.

Results of analyses of two specimens of plagioclase amphibolite from the Eagle mine area are shown in Table IV, cols. *A* and *B*. For comparison, an analysis (Johannsen, ref. 9, p. 305) of an olivine basalt is given in col. *D*. Although Johanis given in col. *D*. Although Johannsen gives a number of analyses of olivine basalt, this particular one was chosen for comparison because its silica content is almost identical with that of the Goldfields plagioclase amphibolite.

This close correspondence in chemical composition would, at first glance, seem to indicate that the plagioclose amphibolite was originally an olivine basalt or equivalent diabase or gabbro. Actually, however, we can do no more than guess at the original composition of the rock, because the extent of metasomatic changes during regional metamorphism in the area is not known.

TABLE III.— ANALYSES AND MINERAL COMPOSITION OF SLATES AND ALLIED ROCKS

	A	B	C	D	E	F
	%	%	%	%	%	%
SiO_2	67.7	57.7	77.8	67.7	56.29	60.88
TiO_2	n.d.	n.d.	n.d.	n.d.	0.64	0.62
Al_2O_3	19.3	22.1	11.3	13.1	19.22	17.78
Fe_2O_3	1.2	3.3	0.6	0.7	4.39	1.94
FeO	0.1	5.6	2.1	4.3		4.07
MnO	Tr.	0.1	Tr.	0.1	—	—
MgO	0.8	4.6	1.3	2.6	1.65	3.53
CaO	0.3	0.8	1.1	1.0	0.09	2.77
Na_2O	0.4	2.0	3.6	3.1	0.19	2.65
K_2O	5.0	4.0	0.6	2.1	10.85	3.16
H_2O	n.d.	n.d.	n.d.	n.d.	5.58	2.04
P_2O_5	n.d.	n.d.	n.d.	n.d.	—	0.29
SO_3	n.d.	n.d.	n.d.	n.d.	0.72	—
C	n.d.	n.d.	n.d.	n.d.	—	1.70
S	n.d.	n.d.	n.d.	n.d.	—	0.10
	94.8	100.2	98.4	94.7	99.62	101.53
F	n.d.	0.035	n.d.	n.d.		
Ga_2O_3	0.0040	0.0043	n.d.	0.0026		
Rb_2O	n.d.	0.061	n.d.	n.d.		
MINERAL COMPOSITION						
Chlorite	—	55.6	66.3	—		
White-mica	45.8	21.6	—	—		
Chlorite & sericite	—	—	—	70.1		
Albite	—	15.1	21.1	8.3		
Quartz	51.5	4.6	2.5	13.8		
Epidote	1.6	2.1	5.1	7.7		
Calcite	—	—	—	Tr.		
Opaque	1.1	1.0	5.0	0.2		

A. —Quartz-muscovite schist, Tazin group, Precambrian; Tamblyn property Goldfields
B. —Chlorite-sericite schist, Tazin group, Precambrian; Ace mine, Goldfields
C. —Brown cherty rock, Tazin group, Precambrian; same location as B
D. —Bedded slate, Tazin group, Precambrian; Ato claims, Goldfields
E. —Glenwood shale, Ordovician, Minneapolis, Minn.; Pettijohn (ref. 10, p. 285)
F. —Knife Lake slate, Archaean, Minnesota; Pettijohn (ref. 10, p. 285)

Plagioclase amphibolites in the Ace mine-area differ in appearance from those on the Eagle property. In the vicinity of the Ace mine, the rock is fine grained, dark greenish-grey, and contains epidote veinlets (1/16 in. to 1/8 in.) in an irregular pattern (crackle breccia).

These veinlets are believed to be due to hydrothermal alteration following brecciation. The epidote does not seem to have been introduced from 'depth' because the veinlets are more or less confined to the plagioclase amphibolite. The mineral is probably a product of reaction between the plagioclase amphibolite and ascending hydrothermal solutions entering incipient fractures from the St. Louis fault zone. A somewhat similar hypothesis has been advanced by Cooke (6) for the origin of chrysotile asbestos veins in serpentine bodies in the Eastern Townships of Quebec.

The composition of the plagioclase amphibolites on the Ace property, and the lack of any evidence of bedding, indicate that they were originally igneous rocks before being metamorphosed. However, as on the Eagle property, there is no field evidence to suggest whether the original rock was extrusive, intrusive, or a complex of both.

An analysis and mineral composition of a specimen of coarse-grained, massive plagioclase amphibolite (amphibolite of Christie and Kesten) from a point on the south shore of Beaverlodge Lake (see Figure 1) is given in Table IV, col. *C*. Christie has indicated on his map that the 'amphibolite' in this locality is of intrusive origin. In comparison with the plagioclase amphibolite of the Eagle mine-area, this rock is slightly more silicic, but on the whole it has a similar chemical composition.

Mafic Inclusions in Alaskite

On the Eagle property, a few chlorite schist inclusions occur in all alaskite bodies mapped. Similar inclusions consisting of amphibolite have been noted by Christie and Kesten (5) in many parts of the Goldfields area. At their margins, the amphibolite inclusions are commonly altered to biotite gneiss.

As seen in drill-core intersections, the chlorite schist inclusions in alaskite in the Eagle mine are speckled white and greyish-green. Microscopic examination shows that they consist almost entirely of chlorite and untwinned oligoclase. Hydrothermal alteration has apparently altered all the original hornblende and/or biotite to chlorite but has not affected the oligoclase. An analysis of a specimen is shown in Table V, col. *B*. The analysis indicates that the chlorite schist is more closely related to plagioclase amphibolite than to any of the other rocks analyzed.

Mafic inclusions in alaskite are common in the vicinity of the Ace mine, south of the St. Louis fault. Generally they are tabular, a few inches to several feet thick, and some are more than ten feet long. They are especially abundant in the contact zone between alaskite and plagioclase amphibolite, where they comprise up to 50 per cent or more of the rock, which then, as a whole, becomes a granite-gneiss.

Close to the St. Louis fault, the inclusions are chlorite schist, but farther south they are plagioclase amphibolite. This suggests that the inclusions near the fault were also plagioclase amphibolite, but were sheared and hydrothermally altered to chlorite schist when faulting took place.

An analysis of a chlorite schist inclusion in alaskite about 50 feet south of the St. Louis fault is shown in Table V, col. *A*. As will be noted, it is almost identical in composition with the chlorite-schist inclusion from the Eagle mine (col. *B*).

Alaskite

Alaskite (or granite) is the most common rock type on the Eagle-Ato-Mic claims. The term alaskite is used in preference to granite because, unlike normal granite, the rock generally contains less than 2 per cent ferromagnesian minerals and is composed almost entirely of quartz and feldspar. The feldspar is commonly albite (An 5-10), but a specimen of alaskite consisting primarily of orthoclase and quartz was collected at a point about 2,000 feet west of Mic lake (Figure 2). The orthoclase- and albite-alaskites are not readily distinguished except in thin section and the areal extent of the orthoclase-bearing type is not known.

A characteristic feature of the alaskite is that it commonly contains 'splashes' or irregular bodies of milky-quartz, up to several feet across.

TABLE IV.— ANALYSES AND MINERAL COMPOSITION OF PLAGIOCLASE AMPHIBOLITES

	A	B	C	D
	%	%	%	%
SiO_2	44.6	44.4	51.6	45.33
TiO_2	n.d.	n.d.	n.d.	2.23
Al_2O_3	16.2	16.2	11.2	14.99
Fe_2O_3	1.7	1.9	3.2	3.38
FeO	10.8	10.6	9.2	8.75
MnO	0.3	0.3	0.2	0.20
MgO	7.0	6.0	4.8	6.93
CaO	5.9	5.2	6.6	8.06
Na_2O	3.6	3.0	3.0	2.94
K_2O	1.6	2.0	1.2	1.00
H_2O	n.d.	n.d.	n.d.	5.66
P_2O_5	n.d.	n.d.	n.d.	0.28
CO_2	n.d.	n.d.	n.d.	0.15
S	n.d.	n.d.	n.d.	0.18
	91.7	89.6	91.0	100.08
F	0.13	0.15	n.d.	
Rb_2O	0.012	0.019	n.d.	
MINERAL COMPOSITION				
Hornblende	52.8	35.1	65.8	
Oligoclase	27.0	49.0	—	
Andesine	—	—	29.2	
Biotite	—	Tr.	—	
Quartz	—	—	1.4	
Apatite	—	0.5	0.6	
Ilmenite	—	—	2.5	
Chlorite	9.4	12.1	—	
Epidote	7.9	—	—	
Calcite	1.0	—	—	
Specularite	1.9	3.2	—	
Hisingerite (?)	—	0.2	0.6	
Pyrrhotite	—	—	Tr.	

A. —Plagioclase amphibolite, Tazin group, Precambrian; Eagle mine-area, Goldfields
B. —Plagioclase amphibolite, Tazin group, Precambrian; Eagle mine-area, Goldfields
C. —Plagioclase amphibolite, Tazin group, Precambrian; South shore, Beaverlodge lake, Goldfields
D.—Olivine-basalt, Livingstone island, Antartica; Johannsen, ref. 9, Vol. 3, p. 305

TABLE V.— ANALYSES AND MINERAL COMPOSITION OF CHLORITE-SCHIST INCLUSIONS IN ALASKITE

	A	B
	%	%
SiO_2	40.0	42.9
TiO_2	n.d.	n.d.
Al_2O_3	21.1	22.7
Fe_2O_3	6.6	5.5
FeO	11.5	14.5
MnO	0.3	0.3
MgO	14.6	9.4
CaO	0.6	0.6
Na_2O	3.8	3.9
K_2O	0.5	0.8
H_2O	n.d.	n.d.
P_2O_5	n.d.	n.d.
CO_2	n.d.	n.d.
	99.0	100.6
F	0.17	0.035
Rb_2O	0.007	0.006
MINERAL COMPOSITION		
Oligoclase	32.4	50.6
Chlorite	61.5	44.2
Calcite	0.9	—
Apatite	—	0.3
Specularite	5.2	4.8

A.— Chlorite-schist inclusion in alaskite, Ace mine-area, Goldfields
B.— Chlorite-schist inclusion in alaskite, Eagle mine, Goldfields

In general, the alaskite is uniform in composition, but locally it grades into granite gneiss showing relict sedimentary bedding.

Alaskite intrudes all of the Tazin rocks described in the preceding pages. Along many of the contacts between alaskite and metamorphosed Tazin rocks, dykelets of alaskite can be seen cutting the latter.

The alaskite is typically salmon-pink, but in a few localities it is white. Generally, the rock is medium to fine grained, and all specimens show evidence of catcalastic metamorphism. On freshly broken surface, feldspar cleavage faces are seldom seen because most of the feldspar crystals have been shattered and the resulting fragments have a haphazard orientation. The finer grained alaskites grade into mylonites which, on fresh surfaces, are not easily distinguished from quartzites. However, the weathered surface of the fine grained alaskite has a chalky outer film up to 1/16 of an inch thick which has a marked kaolin odour. This contrasts with the quartzites, which are vitreous on weathered surfaces.

Where weathered surfaces are not available, the characteristic salmon-pink colour of alaskite serves to distinguish it from quartzite. Invariably, microscopic study has shown that fine grained, cherty looking rocks with a salmon-pink colour are alaskite-mylonites. No quartzites of this colour have been found on the Eagle-Ato-Mic claims.

An analysis of a typical albite alaskite from the Eagle mine area is given in Table VI, col. *B*. For comparison, column *D* of the same Table gives the average of three analyses of alaskite from other localities (Johannsen, ref. 9, Vol. 2, p. 108). As will be noted, the analyses closely correspond except as regards soda and potash, the percentages of which are, of course, dependent on the type of feldspar present.

Two types of alaskite occur on the Ace property, a soda-rich type and a less common potash-rich variety which is confined to a narrow zone north of and parallel with Ace creek. Both rocks are medium grained, cream to salmon-pink in colour, and they are not readily distinguished in hand specimen. The soda-rich alaskite consists primarily of albite and quartz, whereas microcline occurs in the potash variety in place of albite. Analyses of the two types are given in columns *A* and *C* of Table VI. That of the soda-rich alaskite (col. *A*) is almost identical with the analysis of the alaskite from the Eagle property, three miles northwest (col. *B*). The analysis of the microcline-bearing variety also is closely similar, except for relative proportions of K_2O and Na_2O.

TABLE VI.— ANALYSES AND MINERAL COMPOSITION OF ALASKITES

	A	B	C	D
	%	%	%	%
SiO_2	73.5	76.5	72.8	75.05
TiO_2	n.d.	n.d.	n.d.	0.03
Al_2O_3	14.5	13.7	12.8	13.48
Fe_2O_3	0.9	0.8	0.3	0.84
FeO	0.6	0.7	0.8	0.63
MnO	Tr.	0.1	Tr.	0.04
MgO	0.9	0.9	0.6	0.21
CaO	0.5	0.6	0.4	0.24
Na_2O	5.4	5.8	2.2	4.20
K_2O	1.0	0.3	7.5	5.03
H_2O	n.d.	n.d.	n.d.	0.34
P_2O_5	n.d.	n.d.	n.d.	0.08
ZrO_2	n.d.	n.d.	n.d.	0.03
	97.3	99.4	97.4	100.20
F	n.d.	0.001	0.0025	
Rb_2O	n.d.	0.0013	0.019	
MINERAL COMPOSITION				
Albite	57.6	66.9	—	
Microcline	—	—	73.7	
Quartz	30.4	30.3	23.2	
Sericite	10.2	—	1.2	
Calcite	—	0.3	—	
Chlorite	1.5	2.0	1.7	
Epidote	—	0.5	—	
Unidentified opaque	0.2	—	0.4	

A. —Albite-alaskite, Ace mine-area, Goldfields
B. —Albite-alaskite, Eagle mine-area, Goldfields
C. —Microcline-alaskite, Ace Creek, Goldfields
D.—Alaskite, Johannsen; ref. 9, Vol. 2, p. 108 (average of three analyses)

TABLE VII.— ANALYSES AND MINERAL COMPOSITION OF GRANODIORITE

	A	B
	%	%
SiO_2	68.9	67.86
TiO_2	n.d.	0.51
Al_2O_3	19.6	14.97
Fe_2O_3	2.7	1.75
FeO	3.6	2.18
MnO	0.1	0.05
MgO	3.0	1.38
CaO	1.1	2.82
Na_2O	1.5	3.29
K_2O	2.8	3.58
H_2O	n.d.	0.84
P_2O_5	n.d.	0.17
CO_2	n.d.	0.04
	102.9	99.44
MINERAL COMPOSITION		
Albite	20.8	—
Quartz	36.8	—
Green riotite	16.9	—
Chlorite	6.1	—
Epidote	4.2	—
Sericite	12.1	—
Specularite	1.6	—
Pyrite	1.6	—

A. —Gneissic granodiorite, Emar claims, Goldfields
B. —Granodiorite, average of 24 analyses; Johannsen, ref. 9, Vol. 2, p. 331.

On both the Eagle and Ace properties, the alaskites have been mylonitized and, near pitchblende-bearing veins, they have subsequently been further altered. This alteration

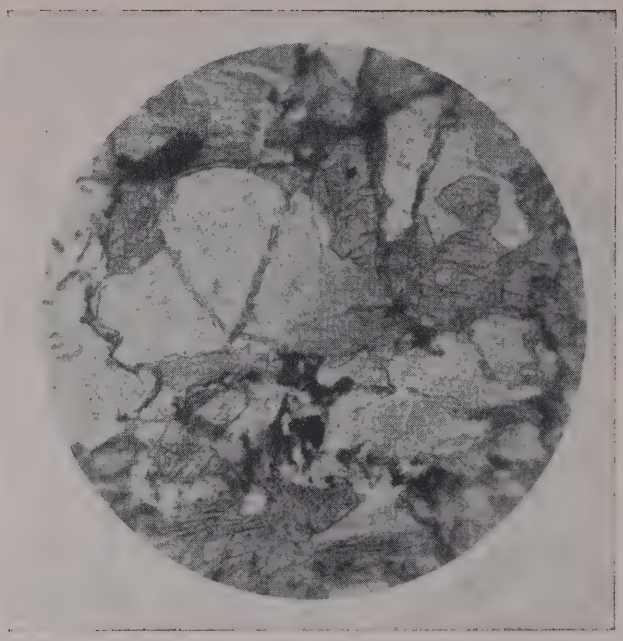


Figure 3.—Thin section of typical plagioclase amphibolite, Goldfields area. ×50.

Oligoclase greyish-white; hornblende dark grey. Ordinary light.

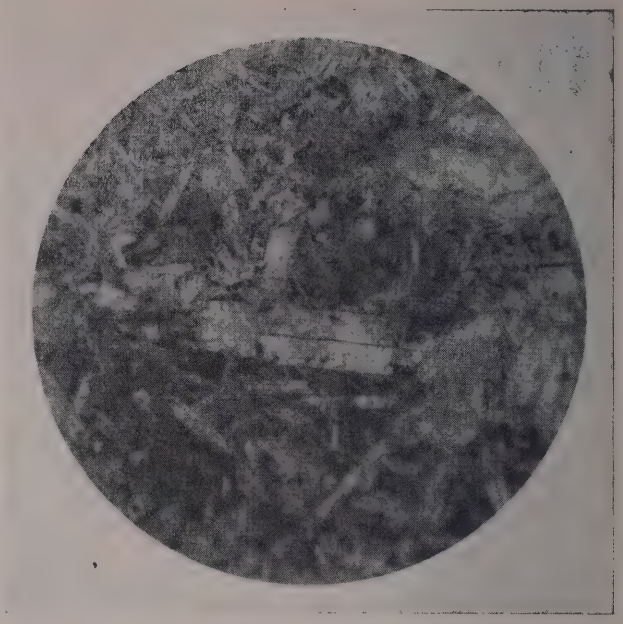


Figure 4.—Thin section of typical diabase, Goldfields area. ×50.

Laths of plagioclase (white) and grains of magnetite (black) in fine grained, indeterminate matrix. Ordinary light.

is largely due to the introduction of finely divided hematite and the final product is a brick-red, cherty looking rock.

Unaltered alaskites at Donaldson lake have a uniform salmon-pink colour and are similar in all respects to the albite alaskites in the Eagle and Ace mine-areas. On the Emar claims of the Eldorado Mining and Refining Company at Donaldson lake, alaskites have been mylonitized, and then subsequently hydrothermally altered and recrystallized. The final product is a rock composed almost entirely of coarsely cleavable red albite and calcite, with minor amounts of quartz, specularite, and chlorite.

Granodiorite

The granodiorite on the Emar claims at Donaldson lake has been altered to a foliated gneiss. It consists chiefly of albite, opalescent quartz, green biotite, muscovite, and chlorite. The rock has a distinctive appearance because some of the albite has been stained orange to red due to varying amounts of hydrothermally introduced hematite. This gives the rock an appearance of woven white and green tweed with a fleck of brighter colours.

An analysis of this rock is shown in Table VII together with an average of twenty-four analyses of granodiorites by Johannsen (9). Except for minor differences, the Donaldson Lake granodiorite has a similar composition to the average given by Johannsen.

TABLE VIII.—ANALYSES AND MINERAL COMPOSITION OF BASALT FLOWS

	A	B	C	D	E
	%	%	%	%	%
SiO_2	49.9	53.7	49.4	52.6	50.31
TiO_2	n.d.	n.d.	n.d.	n.d.	2.25
Al_2O_3	13.2	13.7	14.3	13.0	15.54
Fe_2O_3	9.5	8.1	9.0	2.9	3.09
FeO	2.5	2.8	2.6	7.9	7.72
MnO	0.1	0.1	0.1	0.2	0.21
MgO	4.6	4.7	4.0	3.9	6.67
CaO	3.9	3.0	3.1	6.0	9.50
Na_2O	4.7	4.3	5.2	3.0	2.94
K_2O	2.0	3.0	1.7	4.0	0.68
H_2O	n.d.	n.d.	n.d.	n.d.	0.95
P_2O_5	n.d.	n.d.	n.d.	n.d.	0.24
CO_2	n.d.	n.d.	n.d.	n.d.	0.13
	90.4	93.4	89.4	93.5	100.24
F	n.d.	n.d.	n.d.	0.45	
Rb_2O	0.012	0.021	0.009	0.030	
Li_2O	0.0032	0.0033	0.0038	0.0025	
Cu	n.d.	n.d.	n.d.	0.0044	
Co	n.d.	n.d.	n.d.	0.0013	
Ni	n.d.	n.d.	n.d.	0.0043	
MINERAL COMPOSITION					
Plagioclase	60.2	74.8	29.2	72.3	
Augite	—	—	—	14.1	
Indeterminate matrix	—	—	41.5	—	
Calcite	2.3	Tr.	—	0.4	
Quartz	1.3	Tr.	—	—	
Apatite	—	—	0.9	2.5	
Magnetite	—	—	—	5.7	
Chlorite	11.2	10.9	6.5	5.1	
Epidote	4.9	5.2	4.6	Tr.	
Sericite	2.1	—	—	—	
Specularite	18.1	9.1	17.3	—	

A. —Flow 1, Athabaska series, near adit portal, R.A. claims, Martin Lake, Goldfields
B. —Flow 2, Athabaska series, next flow east of flow 1, R.A. claims, Martin Lake, Goldfields
C. —Flow 3, Athabaska series, next flow east of flow 2, R.A. claims, Martin Lake, Goldfields
D. —Sill (or possibly coarse-grained flow) intercalated in Athabaska series, near southern part of Martin Lake syncline, Martin Lake, Goldfields
E. —Basalt, average of 16 analyses; Johannsen, ref. 9, Vol. 3, p. 261.

Athabaska Conglomerate

Athabaska conglomerate outcrops at several places near the Eagle shaft (Figure 2). The largest body of conglomerate lies along the south side of the Conglomerate fault. The dip of the fault as determined by underground mapping is about 30° to the south. This indicates that the faulting was of the normal type in order to have preserved the Athabaska rocks from erosion. However, the possibility of a graben-type fault slice should not be overlooked.

The Athabaska conglomerate lies unconformably on Tazin rocks and alaskite. It consists of pebbles and boulders up to several feet in diameter of a number of rock types, including at least two varieties of granite and a number of quartzites of contrasting colours. The conglomerate is well indurated and the matrix consists of dark grey, sandy material which in some places appears chloritic and in other localities siliceous. Except where brecciated and highly altered, the Athabaska conglomerate does not present any problem in recognition so that no special petrological study was made of this rock.

Basalt Flows and Diabase Dykes

Basalt flows occur interbedded with arkose and conglomerate in the structural basin at Martin lake. The series is of Athabaska (Late Precambrian) age.

There are numerous diabase dykes in the Goldfields area, occurring both singly and as swarms. Most of them are narrow, from 1 foot to 20 feet wide, and only rarely can they be traced for a distance of more than 100 feet. Nearly all of the dykes mapped by Christie cut rocks of the basement complex where Athabaska rocks have been stripped away by erosion. In general, the dykes strike west to northwest, but a few strike northeast.

Excellent petrographic descriptions of the flows and dykes in the area are given by Christie and Kesten (5) and need not be repeated here. The main point to emphasize is that, in thin section under the microscope, both the flows and the dykes are found to have similar characteristics. Probably the most convincing evidence that these rocks are closely related is the presence in both types of quartz-calcite-chlorite amygdules. Compared with the amygdules in the flows, which are commonly ⅛ to ½ inch in diameter, those in the dykes are small (⅛ inch or less) and they are not readily recognized in the hand specimen. Nevertheless, they have the same composition.

Table IX.— Analyses and Mineral Composition of Diabase Dykes

	A	B	C	D	E
	%	%	%	%	%
SiO_2	53.1	52.7	52.9	51.3	55.1
TiO_2	n.d.	n.d.	n.d.	n.d.	n.d.
Al_2O_3	13.8	17.3	15.2	14.9	14.7
Fe_2O_3	3.3	4.6	1.4	3.4	2.0
FeO	4.8	4.7	6.7	7.7	6.7
MnO	0.1	0.1	0.1	0.1	0.1
MgO	5.4	5.2	6.5	7.6	6.7
CaO	4.2	5.2	4.3	3.7	2.2
Na_2O	2.5	3.8	2.4	2.2	4.3
K_2O	6.3	5.6	1.6	2.0	0.5
H_2O	n.d.	n.d.	n.d.	n.d.	n.d.
P_2O_5	n.d.	n.d.	n.d.	n.d.	n.d.
CO_2	n.d.	n.d.	n.d.	n.d.	n.d.
	93.5	99.2	91.1	92.9	92.3
F	n.d.	n.d.	0.11	n.d.	n.d.
Rb_2O	0.060	0.062	0.009	0.012	0.004
Li_2O	0.0054	0.0049	0.0053	0.0049	0.0032
Cu	0.0015	0.0026	0.0099	0.0058	0.0046
Co	0.0011	0.0010	0.0023	0.0024	0.0026
Ni	0.0037	0.0032	0.0081	0.0073	0.0088
Mineral Composition					
Plagioclase	19.2	—	—	33.6	17.0
Andesine	—	60.6	20.1	—	—
Indeterminate matrix	44.1	—	51.2	30.4	61.4
Biotite	—	17.7	—	—	—
Magnetite	10.2	4.9	—	—	—
Ilmenite	—	—	13.3	6.4	—
Pyrrhotite	—	—	Tr.	—	—
Apatite	0.04	Tr.	—	—	—
Calcite	Tr.	—	—	1.1	—
Quartz	1.2	2.3	0.9	—	0.4
Chlorite	8.8	2.3	4.4	19.1	4.4
Epidote	16.5	12.3	10.2	9.5	16.8
Leucoxene	—	—	Tr.	—	—

A. —Diabase dyke, Ace mine-area, Goldfields
B. —Diabase dyke, Donaldson lake, Emar claims, Goldfields
C. —Diabase dyke, Mic lake, Eagle-Ato-Mic claims, Goldfields
D. —Diabase dyke, Tamblyn group, south shore Beaverlodge lake, Goldfields
E. —Diabase dyke, Strike group, north of Ace lake, Goldfields

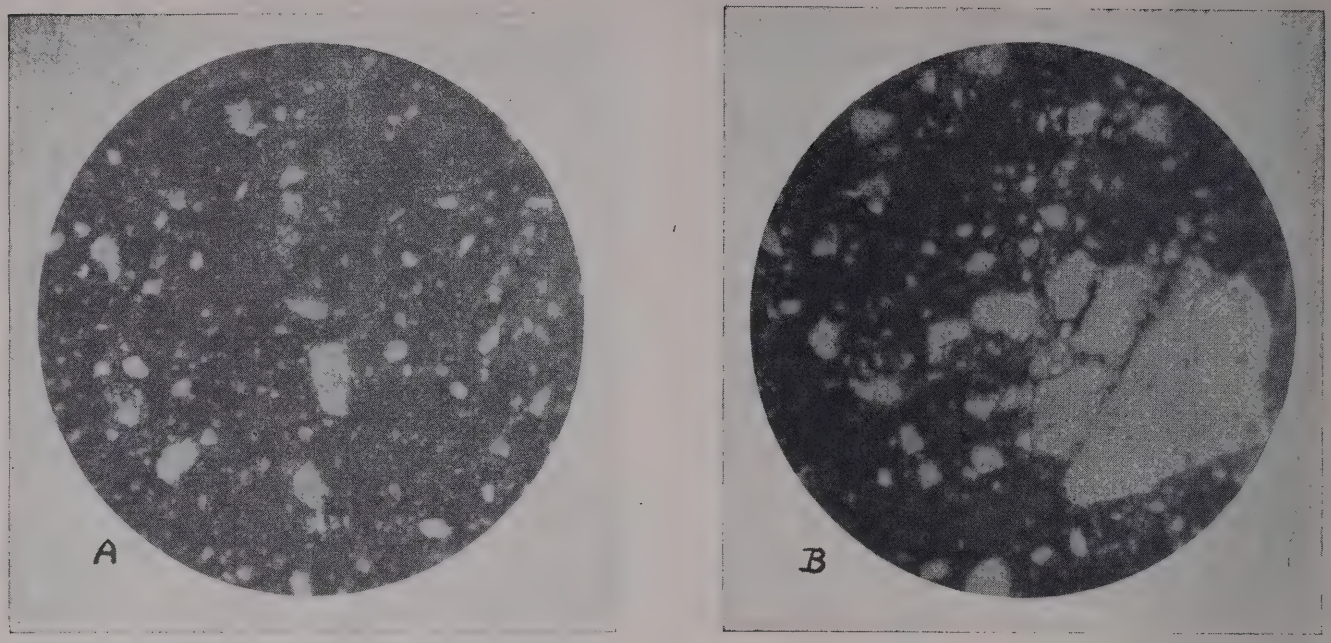
Analyses show that there is a close parallel in chemical composition between the diabase dykes and the basalt flows and sills (see Tables VIII and IX). The greatest difference is the relatively greater amount of ferric iron in the flows. However, this is probably the result of hydrothermal alteration, as the flows contain appreciable amounts of specularite in fractures and as replacement of various minerals.

Structural Breccia

A number of irregularly shaped, discontinuous breccia zones occur on the Eagle-Ato-Mic claims, especially on the northern half of the claim group (see Figure 2). Most of these zones strike northeast. The contacts with other rocks are ragged and, insofar as the author is aware, insufficient core drilling or underground work has been done to determine the overall dip of any one zone.

The fragments in the breccia zones range in size from microscopic mylonitized particles to large, angular blocks many feet across. The proportion of fragments to matrix varies considerably. In some places, the breccia is almost free of recognizable inclusions and closely resembles a black porphyritic trap. However, no phenocrysts are present in the rock and, invariably, white flecks can be shown by microscopic study to be mylonitized fragments of alaskite rather than phenocrysts (see Figure 5). Locally, this uniformly fine-grained black rock grades laterally into typical breccia in which large, angular to subangular fragments of mylonitized alaskite are crowded together and cemented with the same hard, black matrix.

The fragments in any one locality generally consist of a single rock type, but in a few places, as in the breccia complex exposed in the Eagle mine, mixed fragments are found. The most common rock fragment is mylonitized alaskite. Brecciated



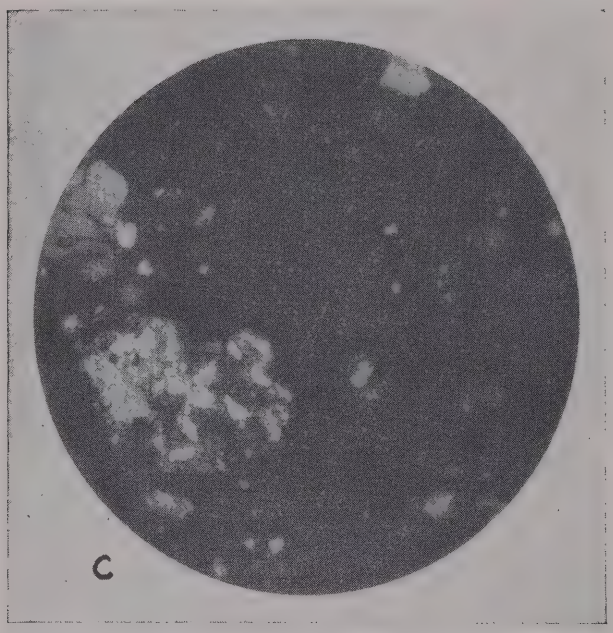


Figure 5.—Photomicrographs of thin sections of black, trap-like mylonite, Eagle-Ato-Mic claims, Goldfields area. *A* and *B*, ordinary light; *C*, crossed nicols. ×50.

Light areas are fragments of mylonitized alaskite; sub-microscopic matrix appears black.

Figure 6.—Sketch of diamond drill core, showing elongated fragments of greywacke-looking rock (chloritized and sericitized alaskite) in black schistose matrix, Eagle mine-area, Goldfields.

(Natural size)

quartzite, plagioclase amphibolite, and Athabaska conglomerate have also been noted in various parts of the area mapped. Brecciated Athabaska conglomerate is particularly difficult to recognize in underground exposures but is more readily identified in diamond drill core intersections and in surface exposures.

The matrix of the breccia is generally dark grey to black and is extremely fine grained. No flow lines were observed and, unlike the Sudbury breccia described by Fairbairn and Robson (8), the matrix is generally massive rather than schistose.

For some time it was believed that the black matrix was trap (or basalt) which had intruded a breccia zone. However, detailed petrographic study with a high-power objective lens, supported by x-ray work, convinced the writer that the matrix was simply a mylonite with hydrothermally introduced chlorite. The same conclusion was reached by E. E. N. Smith (12).

Shand (11) describes a similar breccia zone in South Africa which grades into an extremely fine-grained black rock which he had at first mistaken for basalt. He gave the name 'pseudo-tachylyte' to the black rock in view of its resemblance to tachylyte or basalt glass.

Under the microscope, the breccia matrices studied were found to consist of scattered fragments (1 mm. or less) of mylonitized alaskite in an almost isotropic and nearly opaque groundmass which polarizes in dark greys. The larger fragments exhibit a minute mosaic structure typical of mylonite (see Figure 5). Their margins have a serrated appearance and seem to pass imperceptibly into the groundmass. Some thin sections show a few small clusters of porphyroblastic epidote or clinozoisite crystals which, like

the fragments, fade into the groundmass at their margins.

As the groundmass of the breccia matrix is too fine grained to determine the mineral constituents optically, an x-ray investigation was carried out for this purpose. Powder x-ray photographs of uniformly fine grained black rock showed that it is composed primarily of quartz, feldspar, and chlorite. Small amounts of other minerals are also undoubtedly present, but did not give measurable reflections. As orthoclase and plagioclase have nearly identical *d* spacings for their principal reflections, the composition of the feldspar or feldspars present could not be determined by the method used.

In the Eagle mine, the breccia appears to be a complex. Part of the zone is the common type described in the preceding paragraphs. It consists of angular fragments and blocks of salmon-pink mylonitized alaskite in a hard, black matrix. In some localities, the black matrix material appears to have been injected into irregular fractures in the alaskite.

Exposed along 101 cross-cut south between stations 105 and 107 in the Eagle mine, the matrix is schistose and contains rounded fragments of pebble to boulder size dispersed like plums in a pudding. The rock has been called 'pseudo-conglomerate' by the writer as the rounded fragments consist of white quartz, salmon-pink mylonitized alaskite, and grey quartzite, all of which are found in place in larger bodies at other places in the Eagle mine.

At still other places in the Eagle mine, small sausage or spindle shaped fragments of grey rock in a black schistose matrix are found. In hand specimen, the fragments display the texture, and have the general appearance, of greywacke. They are commonly ¼ to ½ inch in diameter, one to two inches long, and constitute about 50 per cent of the rock. Microscopic evidence indicates that the elongated fragments are chloritized and sericitized albite alaskite. The black schistose matrix has the same mineralogical composition as the lenticular fragments but contains a higher percentage of chlorite. A sketch of a diamond drill-core intersection of this type of rock is shown in Figure 6.

Propylites also are found in schistose parts of the breccia complex. The propylite occurs as lenticular or tabular bodies, commonly many feet in length. In some places nearly flat lying bands of propylite are interlayered with black schistose material carrying isolated rock fragments. The propylite masses undoubtedly were plagioclase amphibolite inclusions in alaskite which were subsequently deformed and hydrothermally altered along with the alaskite.

The structural breccia zone on the Tamblyn property (see Figure 1) is radioactive and is important in this study because it differs in many ways from other structural breccias found in the Goldfields area. The zone strikes northwest and trenching through overburden has exposed a width of at least ten feet. The dip of the zone is not known but is probably parallel with the schistosity, which dips 70 degrees southwest.

In appearance and physical character the breccia rather resembles charcoal. It is dull greyish-black, soft, porous, and friable. The earthy lustre and dark colour are due to finely divided, introduced carbon or carbonaceous material. The rock contains easily recognizable (⅛ to ½ inch long) wisps and lenses of bleached (kaolinized) quartz-mica schist elongated parallel to the schistosity. Joint planes, both parallel and transverse to the schistosity, are stained yellow by limonite. No pitchblende or other primary uranium mineral was exposed at the time of the writer's visit but, in one trench, yellow and orange uranium stain (gummite) was reported*.

*Dr. A. H. Lang, personal communication.

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Quality Control Principles in Coal Preparation*

by J. VISMAN†

(*Annual Western Meeting, Winnipeg, Man., October*, 1952)

(*Transactions, Volume LV*, 1952, pp. 406-411)

SUMMARY

This paper is presented as part of a programme to show the steps taken by industry and others interested in supplying a solid fuel of constant quality to satisfy a consuming market.

The paper deals with the general principles underlying quality control in coal preparation plants, namely, constant quality and close adherance to acceptance limits. It also describes how control of ash content is generally hampered by the probem of coal sampling and the long duration of the ash analysis and how these difficulties may be overcome.

Quality control can be effected in several ways. Thorough blending will reduce variations appreciably, except those of long trend. The latter can be detected by automatic sampling and quick ash analysis. At present, under favourable conditions, a system of half-hourly ash determinations with an accuracy of 0.5 to 1.0 per cent can be realized, to the effect that trends with a 'wave length' of more than two hours can be eliminated within the limits of accuracy of the ash figures. The general solution of the problem is a combination of blending, accurate sampling, quick ash determination, and statistical analysis of the pattern of ash distribution.

The use of statistical methods of analysis, such as the t test and F test, is discussed at some length in order to emphasize the importance of the correct evaluation of observational data if there is to be confidence in the validity of test results.

* * *

General Considerations

COAL, as a fuel, is too cheap for a long haul across the country. The limiting distance depends on the price of the local competitive fuels, and, as the price depends to a great extent on quality, so does quality, to the same extent, command the market.

Many Canadian coals have the qualities essential in a good fuel. The country's reserves are enormous when measured in terms of years' supply and, also, when compared with the reserves of other fuels. Of course, in the early days, the best seams were mined first and the coal mining industry of Canada is — as in other countries — now faced with the increasing need for beneficiation and conservation. Quality, therefore, becomes increasingly important and, in some of the older mines, it is now a vital problem.

One of the desired characteristics of quality is constancy. For those mines operating under marginal conditions, quality control may be the most important factor of all, because it can lengthen the economic life-span of the existing layout.

In other mass-production industries, quality control has been accepted for two or three decades, but the problems were relatively easy. Let us, for a moment, consider a watch factory and observe the manufacturing of a certain part, for instance, the shaft of the balance wheel. These shafts are made on automatic lathes, in mass-production. The operator gauges the diameters of each shaft and accepts only those whose diameters are within certain tolerance limits. So far, he has checked only the quality of the shaft and this is called 'quality testing'. If, during these tests, it appears that one diameter of the shaft increases gradually then this is clearly no chance deviation, but the result of a gradual dislocation or wearing-off of the cutter. The operator now intervenes; he readjusts the cutter, so that the proper diameter is once more acquired. In this manner, needless losses are prevented: the process is under control and this is called 'quality control'.

In coal preparation, the questions concerning quality control, although they are essentially of the same type, cannot be answered as easily as in the watchmaker's factory. First, our product is not an assembly of well-defined units, but a mass of heterogeneous particles of coal contaminated with bone and shale, and produced in an endless stream. Our 'units' are more or less arbitrary and have to be defined before we can start out at all. Such a unit may be a car load, or a day's production, etc.

Second, a problem which, perhaps, has retarded quality control more than anything else is to secure an accurate sample from these units. How is the quality of, say, a car load of coal to be tested, so that deviations can be detected in time, and excessively high and low values be avoided? Opinions on the essentials of coal sampling are still in turmoil and no international agreement has as yet been achieved. It is not surprising, therefore, that quality control in coal preparation has scarcely started, the more so because this subject leads from the realm of certainties into a rather obscure field of probabilities and prognosis. There are no accepted yardsticks here. No measurement of ash content is reliable beyond the first decimal. In many cases, the figures do not even tally within one unit, often as a result of under-sampling. Even when the samples are reliable, they must be analysed quickly enough to detect and prevent an undesirable trend in the quality of the coal.

Briefly, then, accurate sampling, quick analysis, and proper statistical interpretation of observations are indispensable for effective quality control in coal preparation.

The daily production of a certain large coal preparation plant that we will take as an example included, among other products, two thousand tons of coking coal that were sent to two coke plants belonging to the same enterprise. Samples were collected regularly at the preparation plant as well as at the coke plants. A few years after the war, when standards of quality rose gradually to pre-war levels, the analyses made in the three plants were narrowly watched, and sometimes gave rise to lengthy disputes

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in connection with the variability of the product and the error of sampling.

It was generally recognized that the coking coal should be cleaned at a somewhat lower average ash content than the maximum permissible 8.0 per cent in order to prevent too many rejects. The average at that time was 7.0 per cent for the year, which means that 1.0 per cent ash, equivalent to about 1.6 per cent of the production (depending on the washability characteristics) was continuously sacrificed for the sake of quality, and sometimes even this seemed to be insufficient to maintain the permissible standard of 8.0 per cent ash. The Board of the enterprise were dissatisfied with this situation and began an investigation into more economical methods of achieving the standard quality of 8.0 per cent ash.

The Company's statistician indicated that, if the ash content was *never* to overstep 8.0 per cent, then *all* the free ash would have to be cleaned from the coal, because there would always exist some chance of overstepping the 8.0 per cent limit at some time, particularly as there was 2 per cent inherent ash to be reckoned with. Since this was not acceptable to the Board, the statistician added the following suggestion: Suppose the ash content were allowed to overstep the acceptance limit once in twenty times, and a system of quality control were introduced, then the average ash content could be increased about 0.5 per cent, instead of decreased. This suggestion was eventually followed and, as a result, variations were reduced to one-half by means of a system of automatic sampling and quick ash determination which enabled the operator to readjust the machines every half-hour. The average ash content could be raised from 7.0 per cent to 7.6 per cent before the acceptance limit of 8.0 per cent was over-stepped once in twenty times.

The net result was a more constant product, close to the limit of acceptance, and about 1.0 per cent more production. So, instead of acquiring a standardized product at great cost, here a constant quality was produced at a gain of 1.0 per cent in production, which far outweighed the cost of bringing the cleaning process under control.

The example, at the same time, shows the difference between the attitude of the statistician and, for instance, a layman. For the latter, a quantity is wholly defined by one figure. For the statistician, this is not enough; for statistical purposes at least three figures are required; one for the quantity, one for its accuracy, and one for the probability level of this accuracy.

To illustrate the use of the foregoing terms, an example is taken from a daily report of a washery in which the average ash content of the clean coal was 9.78 per cent. For the mine accountant this means that the average ash content was neither 9.77 nor 9.79 per cent but just 9.78 per cent. For the statistician, who knows that the average was found from, say, 16 figures, each possibly with an accuracy of 0.6 per cent and a probability of 99 out of 100, the average is interpreted as $x = 9.78 \pm 0.15$ per cent ($P = 0.99$). This indicates that the actual average ash content may have any value between 9.63 and 9.93 per cent. It is still possible that the actual ash content is below 9.63 per cent or above 9.93 per cent. However, the chance that the ash content lies within these limits is $P = 0.99$, or 99 per cent.

It may well be asked where this system of expressing simple figures in a difficult way may lead. Why is it that a student in the fields of certain sciences can practically dispense with statistics? If the application of these so-called mathematical-statistical techniques is of any use in the down-to-earth business of making coal, what is the reason? The reason is that, in some scientific fields, the variations are usually small and can often be ignored. In coal mining, the variations are so large that they often obscure results of observations — making a clear knowledge of variability a prime necessity.

A day's production of clean coal may have an average ash content of 8 per cent, but during that day the ash may have varied between 3 and 16 per cent. The mine manager will not understand the cause of complaints from his customers until he knows about the variability of his product. Neither can he dispute any claims without being quantitatively informed about the variations that will probably occur. The answer to such problems is 'statistical control'. The sections that follow show how variations are generally described and how the quality limits are predicted.

Measurement of Variations

Quality control is, in fact, nothing but control of variations in the quality of a product. The quality of coal varies as a result of many factors, such as its petrographic constitution, texture of the seam, number of seams mined, the mining system, segregation during transport, the system of cleaning, operation of cleaner, blending, etc. The combined effect of all these factors produces a variability of what may be called a random character. Randomness infers unpredictability and, up to a point, this is true in coal preparation.

Let us, for instance, consider an operation in which raw coal is conveyed by belt from the mine to the raw-coal bin, where it is then loaded into cars. Suppose there are 83 cars of coal that have been accurately sampled, car for car, during a certain length of time. The ash contents, when plotted on a graph, show a zigzag line that may vary from, say, 8.6 per cent to 13.8 per cent ash. No one ash content in this series could have been predicted from the ash figure that preceded it (Figure 1). On the other hand, the experiment shows that the ash of this coal varies between limits of about 8 per cent and 14 per cent and it is, therefore, very unlikely that — all things being equal — ash content of, say, 25 per cent or 30 per cent would occur in the future

This, at least, gives some grip on the matter, but there is one thing more to be said about the observations. It appears that the highest and the lowest values occur less frequently than the values around 11 or 12 per cent. There is only one ash content between 13½ and 14 per cent, and only three observations are found to lie between 8½ and 9 per cent ash. In the middle values, this 'frequency' is much higher, *viz.*, 16 observations between 11 and 11½ per cent ash; 16 between 11½ and 12 per cent; and 10 between 12 and 12½ per cent. Obviously, the chance or 'probability' for the occurrence of high and low ash contents is smaller than the probability of finding an ash content near the average. This probability may be expressed quantitatively. The chance of finding an ash content of 13½ to 14 per cent is only 1 in 83 (the total number of observations in this example), or 0.012, whereas the probability for the occurrence of an ash content of 11½ to 12 per cent is 16 in 83, or 0.192. This decimal expression for probability is called the 'relative frequency' of the ash content.

The behaviour of a variable quantity can be predicted from its frequency distribution, as long as the circumstances under which the test was performed do not change significantly. The laws of probability are founded on the universal phenomenon that the relative frequency decreases with increasing deviation

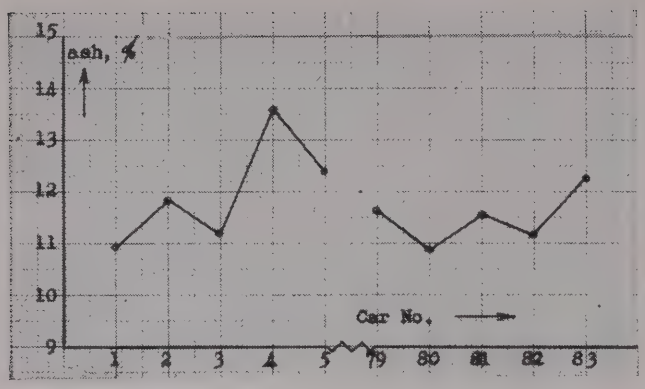

Figure 1.—Ash content of raw coal.
(Successive carloads)

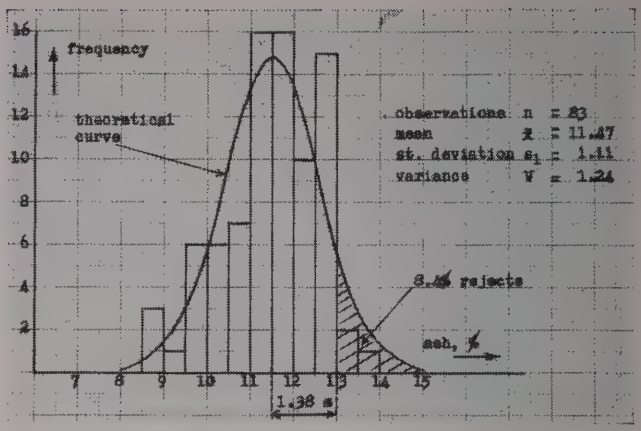

Figure 2.—Frequency diagram of raw coal ash.

Note: In the text, X has been used to designate the 'mean' value, shown in Figures 2, 3, and 4 as small x with line above.

from the mean. Experience has taught, however, that significant changes in the circumstances are more likely to happen than is expected by some statisticians. It is perhaps this experience that made George Canning exclaim, "Statistics? I can prove anything by statistics, except the truth!!"

Statistics, therefore, may perhaps be defined as the formal logic of probability judgments. It is a formal truth. The statistician, in order to get really useful results from his observations, will be well advised to work hand-in-hand with the engineer who is conversant with the vagaries of coal mining and coal preparation. For, although mathematical statistics is a science, its application in the coal mining industry is still much of an art.

If the 83 ash contents mentioned above were classified in class intervals of 0.5 per cent ash, and if the number of cars in each class interval were plotted against the corresponding ash contents, then the curve delineating the contour of the bars representing the frequencies would show the typical shape of a church bell (Figure 2).

What are the limits of the frequency curve? Theoretically, the curve would come nearer to the X-axis the farther it is removed from the mean, but it would never cut that axis. This means that, theoretically, there is always a chance for the occurrence of ash contents higher than 13.8 per cent or lower than 8.6 per cent if the test were to be carried on indefinitely, but the chances for these extreme values are very slender.

The test on the raw coal was repeated four months later. It then appeared that two observations out of the 106 made were lying within the interval 14—14½ per cent ash, whereas none had been found in the first test. Also, one ash content occurred in the interval 8—8½ per cent, whereas none had occurred in this class before (Figure 3). The maximum deviation in this case, therefore, is somewhat larger, although the two frequency distributions are, on the whole, very much the same.

A proper way to express the variation in ash would perhaps be achieved by finding the average deviation of the single observations from the mean ash content. This has indeed been the practice for a long time. It has the advantage that all observations contribute to the accuracy of the result. On the other hand, when the variability is expressed simply by the range

$$w = 13.8 - 8.6 = 5.2\%$$

with only two observations being used, the result is less reliable.

In the examples under review, the frequency-curves found from the tests are based on 83 and 106 observations, respectively. As such, they are still only an approximation to the ideal curve that would have been found if the number of observations had been increased infinitely. The imperfection of each frequency distribution is shown by small peaks and gaps in the bell-shaped curve. It is clear that the average deviation will not be exact, and that its accuracy depends on the number of observations. Results obtained by this method might sometimes be embarrassing if deviations were found which were larger than could be expected from the average deviation. It is the large deviations that give rise to disputes in connection with guarantees.

In course of time, a second yardstick, known as the standard deviation, has been accepted as more reliable. It is the 'root mean square of deviations', given by the formula

$$s^2 = \frac{\Sigma(x_i - X)^2}{n-1}$$

where

s = standard deviation of single observations

x_i = the i^{th} observation; i = 1, 2, 3 . . . n

X = average of (n) observations

n = number of observations

Large deviations affect the standard deviation more than smaller deviations, because they occur in the squared form, as shown by the formula. The standard deviation, therefore, is a more sensitive estimate of the variability than the average deviation mentioned above. The standard deviation and its square, which is called 'variance', are the two quantities generally used for the measurement of variability.

To show the use of the standard deviation, the 83 observations on the ash content of the raw coal will be considered again. The mean ash content appears to be x = 11.47 per cent. A practical question that may arise in this connection is the following: suppose the coal is to be sold on the guarantee that the ash content will not surpass 13 per cent, what percentage of the coal would have to be rejected?

There are two ways of answering this question. The first one is very simple. Among the 83 observations there are three surpassing 13 per cent ash, as shown in Figure 2. The number of observations lying over 13.0 per cent therefore is 3 out of 83, or 0.036 (relative frequency). The chance of finding a car with over 13 per cent ash would, according to this figure, be 3.6 per cent — indicating the percentage of cars that would have to be rejected.

The second way of calculating the percentage of rejects involves the computation of the standard deviation. This has the advantage, that *all* the observations contribute to the accuracy of the result, instead of only three, and that it is sensitive to large deviations. According to the formula, the standard deviation, s, equal 1.11 per cent. Suppose, now, the frequency curve of the observa-

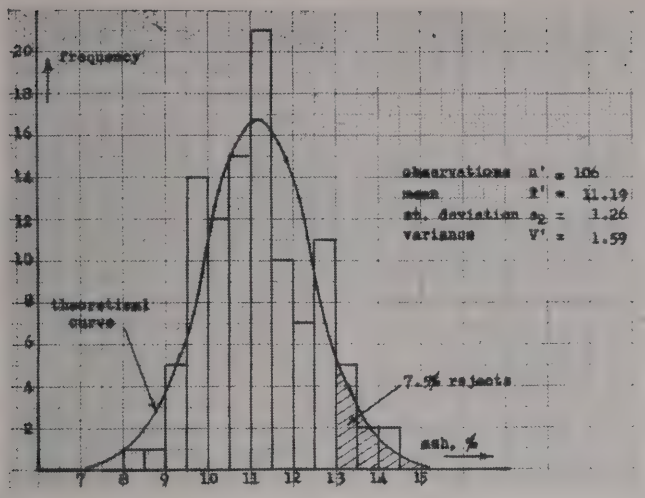

Figure 3.—Ash frequency diagram of same raw coal (Figure 2) four months later.

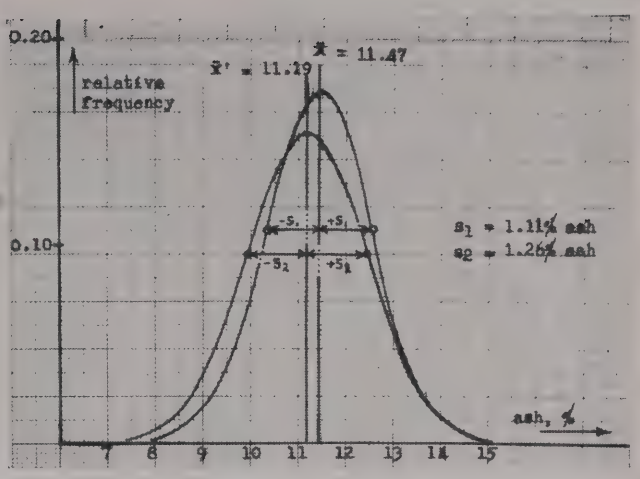

Figure 4.—Comparison between two frequency curves.

tions had the ideal symmetrical bell-shape of the probability curve, then the area under the curve, between the limits $(x - s)$ and $(x + s)$, i.e., between 10.36 per cent and 12.58 per cent ash, would, according to theory, be 68 per cent of the total area. The percentage actually found was 64 per cent.

Also it is possible to calculate these percentages for any portion or multiple value of the standard deviation. The percentage of observations lying between, for instance, $(X - 2s)$ and $(X + 2s)$ is 95.4 per cent, and the percentage lying at the left hand side of $(X + 2s)$ is theoretically 97.7 per cent. Every handbook on statistics (1) contains a table stating these percentages, and from it the answer to the above question, *i.e.*, the percentage of rejects, may be found as follows (Figure 2): the mean ash content of the coal, 11.47 per cent, differs by 1.53 per cent from the 13.0 per cent guarantee. This difference d, expressed in terms of the standard deviation (s = 1.11 per cent), equals $1.53s/1.11$, or $1.38s$. According to the table, this value corresponds with a relative frequency of 1,000 — 0.832/2 = 0.084, or 8.4 per cent. Thus, *all things remaining equal*, the number of cars with more than 13.0 per cent ash may — in the future — be estimated at 8.4 per cent; which is a higher, and also a safer, estimate than the 3.6 per cent found by the first method.

But conditions change in a mine and one is never certain whether the reject percentage still applies; both the mean ash content and the variability may have changed. The only thing to do is run another test series and then, by statistical methods, compare the results with those of the first test series. Methods have been developed to show whether or not any differences are due to chance deviation or significant change.

In order to illustrate these methods, the first and second test series, of 83 and 106 car loads of coal respectively, may once more serve as examples. There are two things to consider in estimating the percentage of rejects. It is important to know, first, whether the *mean ash content* of the coal has changed significantly since the first test was carried out, and second, whether the *variability* of the ash content has changed significantly.

In the second test series, run four months after the first, 106 car loads were tested; the mean ash content was 11.1 per cent and the standard deviation was 1.26 (Figure 3).

Let us suppose that, at the time when the second test series was performed, the coal was still mined from the same four seams as when the first test was made. Two of these seams are pretty dirty, and let us suppose that the proportions of the various coals in the mine-run are somewhat different. Our first problem is to find out whether the mean ash content has changed materially, so that the percentage of rejects is no longer the same.

By use of the '*t*-test' we can determine whether or not the difference in average ash content in the two tests is due to chance or indicates a significant change. This is done by comparing the difference with its own standard deviation.

The t-Test

By means of this test it can be proved whether the difference between two mean values of a quantity is significant. The test is performed with the following formula:

$$t = \frac{X - X^1}{s_t}$$

where X, X' represent the two mean values. s_t is the standard deviation of this difference and is derived from the standard deviation of X and X^1.

The first step therefore is to calculate the standard deviations of the means. According to theory, this quantity is $1/n$ times the standard deviation of a single observation so, for the first test series, the standard deviation of the mean is

$$s' \text{ (mean)} = 1.11/83 = 0.122\%$$

and for the second test series it is

$$s'' \text{(mean)} = 1.26/106 = 0.123\%$$

The problem now resolves itself to the question whether or not the *difference* between the two averages, $X - X^1 = 11.47 - 11.19 = 0.28$, can be explained as a result of *chance deviation* of the average ash contents (Figure 4).

The value of the standard deviation (s_t) is calculated from a formula that will not be given here (1, p. 58). It is found that $s_t = 0.177$ per cent ash. The difference $(X - X^1)$ is now compared with s_t. A table giving different values of the ratio t has been developed (1, p. 270) and the significance of the difference may thus be found. In the above case, the ratio is $(11.47 - 11.19)/0.177 = 1.58$. From the t-table, it is found that the difference between the two averages can be neglected, the critical value of t being 2.61.

The conclusion, therefore, is that the average ash content of the coal

(1) For references see end of paper.

has not changed materially during the interval between the two tests.*

The F Test

The second question that has to be answered with regard to the reject percentage is whether the variability of the ash has changed significantly during the period between the first and the second test series. The F test is used for this purpose.

This test is used to judge the difference between two variances. It is performed with the formula $F = V_1/V_2$, where V_1 is the largest variance and V_2 the smallest, variance being defined as the square of the standard deviation (s^2). F is called the F ratio.

As in the t-test, the Null Hypothesis is used, it being assumed that the variances do not differ significantly.

In our case, the largest variance of single observations is $s_2^2 = 1.59$ and the smallest is $s_1^2 = 1.24$. The ratio $F = 1.59/1.24 = 1.28$. According to the F table, the significance of this ratio is within the limits of chance deviations and therefore may be ignored.

The reader will, at once, realize the importance of these two statistical tools — the t test and the F test. They set the mind at rest about the validity of the test results and enable operators to make predictions with confidence. All questions concerning quality control demand such correct evaluation of observational data. In the next section it will be shown how quality control can be achieved with the help of statistical methods.

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*The Null Hypothesis: The comparison of two means is essentially simple. The means are first assumed not to differ significantly. This is called the Null Hypothesis. The difference may be large or small. When the difference is large, the Null Hypothesis may be confidently rejected. This means that, with a large difference between the two averages, there is only a very slender chance that they are identical. If, however, the ratio is small, it is not so simple. The difficulty lies in the fact that the mean values may differ slightly, yet significantly. The relative inaccuracy of the means, however, makes it impossible to detect such small significant differences. It is important to understand this 'unilateral' character of the Null Hypothesis. A difference between means can be proved to lie within or outside the limits of chance deviations, but the statistical significance of this difference can only be proved in the latter case, or as O. L. Davies states: "a Null Hypothesis may be 'disproved' but never proved" (1, p. 53).

Quality Control

From the calculations in the previous section, for the first test series, it appeared that the percentage of cars containing more than 13.0 per cent ash was 8.4 per cent. From the second test series, the number of rejected cars, calculated in the same way, would equal 7.5 per cent. These percentages are high and it is understandable that the cost of unloading them is considerable. Is it economical to diminish the variability of the coal by blending it to such a degree that not more than 1 per cent of the cars needs to be rejected? This is partly a financial problem which cannot be solved here, but the statistician can give an accurate estimate of the minimum number and volume of bins that are necessary to fulfill this condition, as follows:

The average ash content (2 test series), $X = 11.3$ per cent.

The average standard deviation (2 test series), $s = 1.20$ per cent.

The difference, $(13.00 - X)$, or 1.67 per cent, should, according to theory (1), correspond with 2.32 times the new standard deviation after blending, according to the demand that only 1 per cent of the cars may be rejected:

$$1.67 = 2.32\, s'$$
$$s' = 0.720$$

The required standard deviation (s'), therefore, is 0.6 times the existing standard deviation (s). The number of bins can now be calculated.

Suppose that two bins were used for blending in equal proportions, the new variance (s'^2) would be half the original variance; or the new standard deviation (s') would be only 0.71 times the original standard deviation (s). Therefore, two bins would not suffice; three bins would be needed to fulfill the condition. The statistician, seeing that two bins are nearly enough, calculates the percentage of rejects for both cases. He finds that, for two bins, the reject percentage would be 2.5 per cent, whereas for three bins it would be only 0.8 per cent. The volume of each bin should measure at least one car load. It is understood that the bins are emptied simultaneously and at the same rate, on a belt or chute, so as to ensure proper blending. The most economical solution can then be found from a cost analysis.

Blending is only one method of obtaining control of variations in the quality of the coal. It is by far the simplest method and its possibilities should be exploited in every mine. But blending has its limitations. A mine with 1,000 tons a day capacity cannot blend the coal of the morning with that of the afternoon shift if its total bin capacity is only 500 tons. Yet, the coal produced during the morning may, on the average, differ in ash content and ash distribution from the afternoon coal, as a result of a different cycle of operations.

The only way to acquire constant quality in this case is through timely readjustment of the preparation machines. Several articles (2, 3, 4) have been published on this subject that deal quantitatively with the technicalities of sampling and quality control. Only the main line of thought will be given here, to illustrate the possibilities, and also the limitations, of quality control in coal preparation.

The first and most important function of quality control in the preparation plant is to ensure the proper functioning of the washers. This is relatively easy and is mostly done by regular sink-and-float tests of both the clean product and the refuse of the machines, preferably in carbon tetrachloride. Many manufacturers guarantee coal cleaners on this basis, and experience has shown that the percentage of 'sinks' in the clean coal and of 'floats' in the refuse afford a highly valuable means for spotting maladjustment and mechanical trouble in the washer.

But control of the finished product, not control of the machines, is the ultimate aim of quality control. The cleaned coal, leaving a washer that is perfectly under control, may vary appreciably in ash content as a result of variations in the ash constitution of the raw coal.

Let us suppose that a mine producing this kind of varying run-mine coal has a daily output of 8,000 tons. Complete blending before preparation would require an enormous bin capacity. In such a plant, a system of quality control can be introduced for the cleaning of the fines, in order to reduce the variations in ash of the cleaned product. The operator is provided with a control chart and the average ash content of the cleaned product is marked on this chart every half hour by an analyst.

The effectiveness of such a system depends on the accuracy of the half-hourly samples. Generally, a rise or fall of, say, 1.5 per cent in the average ash content during one half-hour can be detected only if the accuracy of the samples is at

least about 1.0 per cent ash, *i.e.*, roughly two-thirds of this difference as a minimum. The accuracy of sampling a heterogeneous coal can be ensured only if a sufficiently large number of increments are collected. This should be done by automatic samplers which can collect large numbers of small increments (2, 3).

The effectiveness of the system also depends on how quickly the ash content of the sample can be determined. The standard method of ash analysis requires two hours. Obviously, this method is too slow because the ash content may change considerably during that time. A very quick method is to estimate the ash content by means of *X*-ray absorption; but the accuracy of this method is relatively low because it depends on the variability of the chemical composition of the ash and of the coal substance. The best available method is the direct analysis of ash content by means of oxygen. It is possible to dry, grind, and analyse the sample in 20 minutes, with an accuracy of 0.3 per cent ash ($P = 0.99$), and a total accuracy (including sampling) of 0.7 to 0.8 per cent ash. This method enables the operator to detect and eliminate the more persistent changes in ash percentage that may affect the average ash content of the shift production. Every time-series of ash contents (Figure 1) can be regarded as the result of a series of elementary waves, differing in wave length and amplitude (2). A system of half-hourly ash determinations enables the operator to eliminate (within the limits of the accuracy of the ash figures) all the waves with a wave length of more than two hours.

J. M. Juran (5) states that, generally speaking, half the variations can be eliminated by introducing quality control. This appears to hold true for quality control in coal preparation generally, provided the operator's interest is secured by a premium that is inversely proportional to the variability of the finished product.

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Hydraulic Transportation of Oil-Sand Tailings in Small-Diameter Pipes

A Laboratory Scale Study

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ABSTRACT

Laboratory-scale studies were made to investigate the flow characteristics of sand in connection with hydraulic tailings disposal at projected Athabaska oil-sand workings. Friction factor determinations showed that a critical velocity for pipeline flow existed, in agreement with the observations of other investigators; the magnitude of the critical velocity as influenced by various operating variables was measured. The effect of temperature was not great. Increasing the temperature from 100°F. to 160°F. brought about a change in critical velocity from 8.5 ft. per sec. to 8.1 ft. per sec. The effect of solids concentration was much more pronounced with a change from 25 per cent to 49 per cent solids bringing about a change in critical velocity from 6.6 ft. per sec. to 8.8 ft. per sec. Pipe size had an appreciable effect on the critical velocity. For a 50% solids suspension, the critical velocity was 8.4 ft. per sec. in a two-inch pipe and only 6.9 ft. per sec. in a one-inch pipe.

Introduction

ANY PLANT processing mined Athabaska oil sand for the recovery of oil produces, as a waste material, about one ton of sand for every barrel of oil recovered. A disposal problem of major proportions is thus indicated. For a 100,000-barrel per day plant, provision must be made for moving about 4,000 tons of sand per hour on a continuous basis to a tailings dump at least several thousand feet removed from the plant site.

Of the many known ways of conveying solids on a large scale, the hydraulic pipeline transfer method is becoming increasingly important because of convenience and economy of operation. It has been in use for many years in mill operations at metalliferrous mines for the disposal of tailings and as such was a natural outgrowth of hydraulic sluicing (3). Pipeline transportation of mill concentrates has been adopted recently (1), wherein high-density solids are moved several miles by hydraulic pipeline. The flotation power of the water suspending the solids may be increased by addition of magnetite for the handling of ore in pipelines (8). Low-density solids such as coal have received considerable study for low-cost pipelining (2) and a successful operation with coal has been carried out in England with indications of considerable savings over other methods of transportation (9). Dredge operations are based on pipeline transportation for the dredged solids, and the scale of pipelining there has involved pipe sizes as large as 32-inch (6). However, the tendency has been to regard the pumping of solids in water merely as a simple method of transportation and little consideration has been given in the published literature to study of the most economical modes of operating a pipeline.

The chief difference between operating a suspended-solids pipeline and a conventional homogeneous-fluid pipeline is that consideration must be given to the instability of the solids suspension. The solids are kept in suspension by the turbulence of the flow and, if the turbulence is insufficient, the solids come out of suspension and tend to block the pipeline. Although it would appear advisable to operate a given pipeline in such a manner that no settling occurred, at least one well-known tailings pipeline is operating under settling conditions — that of Climax Molybdenum at Climax, Colorado (7). The Climax pipeline operates by gravity flow where the question of pumping power is not important. For the general case in which solids are pumped considerable distances in a horizontal pipeline, the cost of power becomes important and much study must be given to the effect of the operating variables on the power demand. It is to this end that the present investigation is concerned with the manner in which the friction factor of a sand suspension in a pipeline varies with the rate of flow, concentration of sand, temperature, and pipe size.

Experimental Procedure

The sand used in this investigation was obtained from the sand tailings dump of the oil-sand separation plant of the Alberta government, at Bitumount. It had a residual oil content of about 0.3 per cent. Table I shows the size distribution of the ignited sand particles, and Figure 1 is a photomicrograph of the sand particles.

Table I.— Size Analysis of Bitumount Sand

Mesh Size	Yield, wt. %
+30	0
− 30+50	19.7
− 50+80	52.3
− 80+100	15.6
−100+200	11.6
−200	0.8

The friction factor for the flow of sand suspensions in pipelines was determined in the usual manner by measuring the pressure drop along a length of horizontal pipe conducting the flow of material under study. The sand was suspended in water in the reservoir of the pumping assembly and was pumped from there to the measuring pipes by a centrifugal water pump. The scale of operations selected for study was as large as could be used satisfactorily in the laboratory. Thus the pipe size limit was two-inch diameter, and flows ranged from 10 to 110 Imperial gallons per minute. The

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(3) For references, see end of paper.

parallel to it and discharged into the pump reservoir. The pump discharge entered one horizontal length of test pipe and returned by another lying parallel to it. In the case of steel pipes, the pressure drop or head loss was measured between stations fourteen feet apart in the downstream part of the twenty-two-foot lengths of pipe; for the brass pipe the stations were ten feet apart in a sixteen-foot length of pipe. Each station consisted of a brass block machined to fit the contour of the pipe and was drilled and tapped to take an ordinary ¼-in. fitting. The block was welded to the pipe and communication between the well in the block and the inside of the pipe was established by an ⅛-in. hole drilled through the wall of the pipe. Care was exercised that no obstruction resulting from the drilling was left inside the pipe. Each station was fitted with an air trap leading to an inverted differential manometer.

Flow rates of the desired magnitude were established by adjusting a by-pass on the pump discharge and by constricting the discharge opening of the line of test pipe. Flow rates were measured by timing the accumulation of a measured volume of suspension in a receiver set above the pump reservoir. Specific gravity of the suspension was calculated from the weight of a known volume of discharge. From the specific gravity was calculated the solids content of the suspension, assuming a specific gravity of 2.65 for the sand. The precision of the measurement of flow rates and specific gravity was ±1 per cent and ±0.3 per cent respectively. The precision for the pressure measurements was about ±5 per cent. Flow temperatures above room temperature were attained by the injection of low-pressure steam into the pump reservoir. Temperature of the flow-line discharge was taken as the operating temperature.

Flow measurements were made under various conditions of solids concentration, temperatures, pipe sizes, and pipe materials. The friction factor, f, was calculated by means of the formula:

$$f = g \times DH/2LV^2$$

where

g = Acceleration due to gravity in ft. per sec.^{2}

D = Pipe diameter in ft.

H = Loss of head over L ft. of pipe, in ft. of water.

V = Mean space velocity of suspension in the pipe in ft. per sec.

Mean space velocity was calculated from the volume and diameter of pipe.

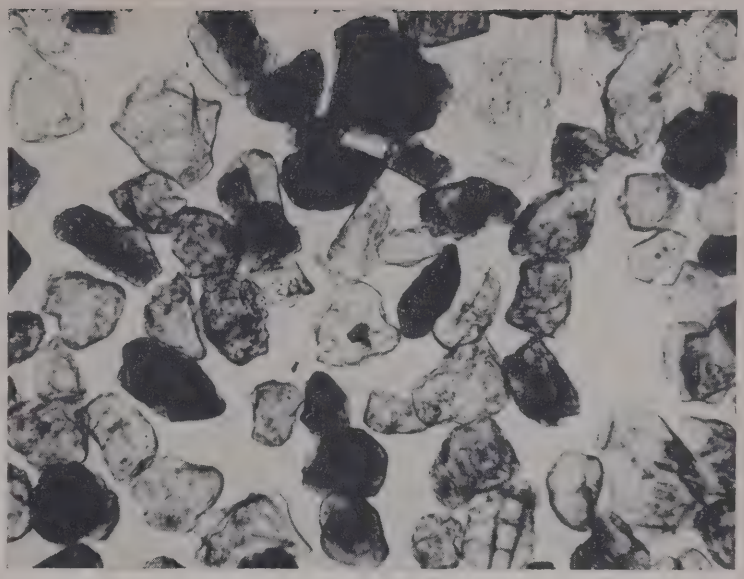

Figure 1.—Photomicrograph of Bitumount sand particles. × 60.

Results

Figure 2 shows two typical curves of friction factor plotted against mean space velocity. The solid curves labelled 'steel' and 'brass' are for clear water. The experimental points are for a sand suspension and they show that, at high velocities, the observed friction factor for a sand suspension is essentially the same as that for clear water, while in flows below a certain critical velocity the observed friction factor is very much greater than that for clear water. The incorporation of a length of glass pipe into the flow line confirmed observations of other investigators (4) that the abrupt change in the friction curve was a result of solids settling out of the suspension and accumulating in the pipe, thus restricting the flow. It is reasonable to extrapolate the steep portion of the curve to the clear-water curve and designate the velocity at the point of intersection as the 'critical velocity' for the given suspension. In this manner, the critical velocity is defined as the mean space velocity of the suspension in the pipe at which accumulation of solids in the pipe begins to occur.

The settling of solids from suspension in a pipeline is a well-known phenomenon, but the mechanism of the process and the controlling factors are not well defined. Some investigators have approached the problem from a consideration of the stability of the suspension as influenced by the conditions of flow. O'Brien and Folsom discussed the velocity component distribution in a turbulent stream and reached the conclusion that the mean space velocity in the pipeline should be about six or seven times the mean free fall velocity of the particles being conveyed (6). This calculation was based on the measurements of fluctuations in velocity made both with an ultramicroscope and with a hot-wire anemometer. Within the turbulent central core, the velocity was believed to be independent of viscosity and therefore independent of temperature.

Another approach to the matter of instability of a suspension of inert solids was made by Wilson (7). The analogy is drawn between the carrying of solids in a pipeline and the picking up of solids from the bed of a stream — the bed-load problem. Investigations made on the bed-load problem indicated that, if the ratio

$$\frac{V_s}{\sqrt{\pi_0/\rho}} \quad \ldots\ldots\ldots\ldots\ldots\ldots (1)$$

(in which V_s is the settling velocity of the solids, π_0 is the bottom shearing stress per unit area, and ρ is the unit density of the liquid) is equal to or greater than one, little or none of the material with settling velocity V_s will be placed in suspension (5). Wilson showed that the condition for the settlement of solids should therefore be

$$\frac{V_s}{\sqrt{HgR/L}} = 1 \ldots\ldots\ldots\ldots (2)$$

where H/L is the head loss, and R is the hydraulic radius of the pipeline. In terms of observed friction factor, (2) becomes

$$V/V_s = \sqrt{2/f} \ldots\ldots\ldots\ldots (3)$$

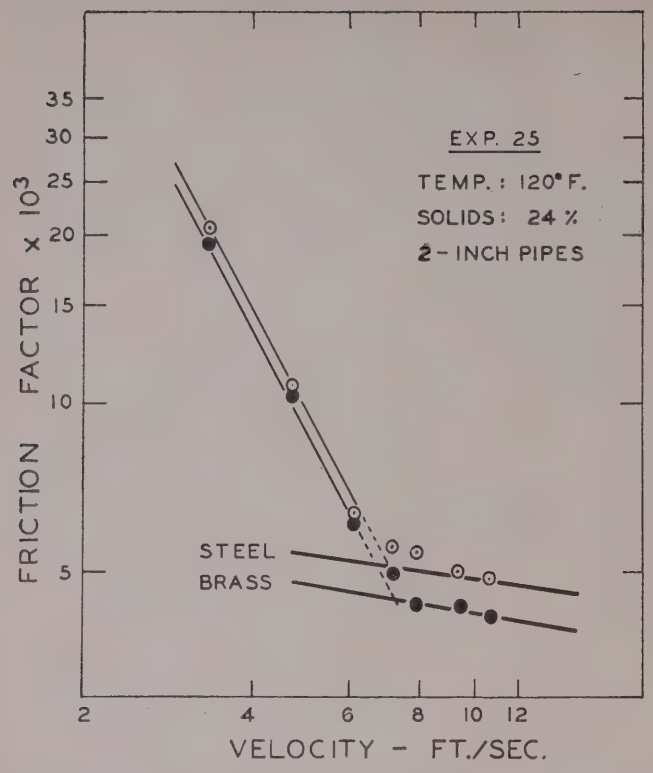

Figure 2.—Typical curves showing relation between friction factor and velocity of a sand suspension in a pipeline. Lines labelled 'steel' and 'brass' represent the relation between the friction factor and velocity for clear water in these pipes. The experimental points are for the sand suspension in the two pipes, and the velocity at which the 'suspension' curves join the 'clear water' curves is termed the 'critical velocity'.

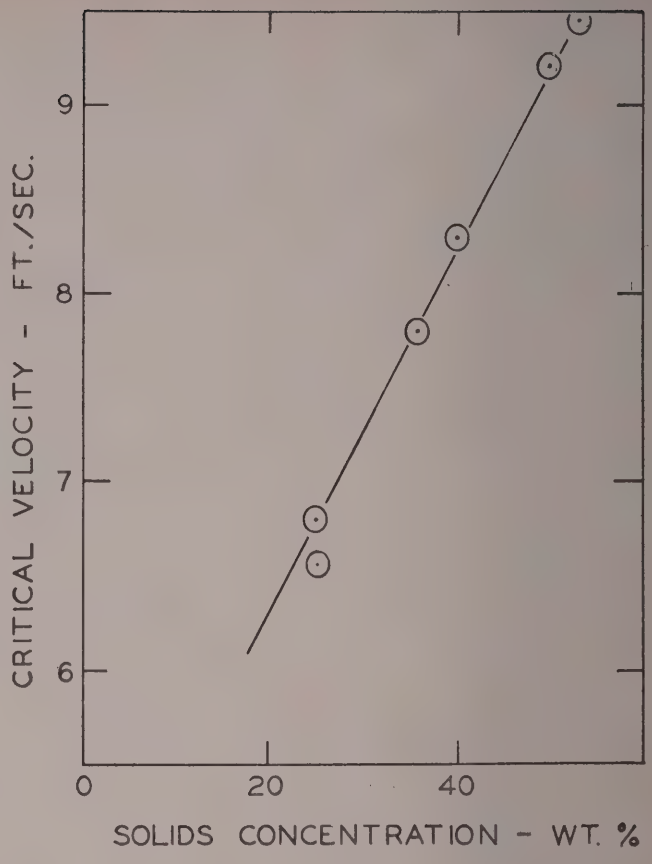

Figure 3.—Effect of solids concentration on critical velocity of sand suspensions. Data are for flows at 120°F. in a 2-inch steel pipe.

The critical velocity has been defined as the velocity at which the observed friction factor no longer coincides with that for clear water. In general, as shown in Figure 2, this occurs in the present study at a friction factor of about 0.005. The value of the V/V_s ratio would therefore be expected to be about 20. Obviously this is not in good agreement with the value of 6 or 7 indicated above.

To test the appropriateness of the foregoing values for the V/V_s ratio, the free settling rates for the sand used in the present investigation were determined. The results are shown in Table II for sized fractions of the Bitumount tailing sand. The settling rate was determined in the usual manner by timing the fall of the sand particles in a column of water.

If the settling rate of the −30 to +50 mesh fraction (making up 20 per cent of the total) of the sand aggregate is taken as being representative of the settling action of the aggregate insofar as clogging of the pipeline is concerned, the ratio becomes $V/0.22$ for a flow temperature of about 120°F.

Figure 2 indicates a critical velocity of 7 ft. per sec. for a 24%-solids suspension in a two-inch pipe at 120°F. The velocity ratio for this system is therefore about 30. While this is not in good agreement with the bed-load prediction, the agreement is better than with the ratio suggested by O'Brien and Folsom.

The relation between critical velocity and solids concentration shown in Figure 3 indicates that it is unsound to rely upon a critical velocity from an experiment selected at random, because of the effect of solids on the critical velocity. To obviate the great effects of solids concentration in the suspension on the critical velocity, the straight line drawn through the points in Figure 3 should be extrapolated to zero concentration. In this manner, the condition of free settling is approached, and testing of equation (3), in which the free settling velocity of the particle is involved, will be more valid. The extrapolated value of the critical velocity so obtained is about 4.2 ft./sec. The ratio of the velocities then becomes $4.2/0.22 = 19$, in good agreement with the solution of the bed-load problem. Hence there is good reason to believe that a fundamental consideration in the hydraulic transportation of solids in a pipeline is the shearing effect of the flowing liquid at the bottom of the pipe as it affects the pick-up of solids, rather than its contribution to the turbulence of the stream as a whole.

TABLE II.—SETTLING RATE OF BITUMOUNT SAND IN WATER

MESH SIZE	FALL VELOCITY, ft./sec.	
	80°F.	175°F.
− 30+50..	0.185	0.265
− 50+80..	0.087	0.136
− 80+100..	0.067	0.104
−100+200..	0.045	0.065
−200.....	0.018	0.026

Referring again to Figure 3, the reason for the positive slope of the curve is not immediately apparent if it is only a matter of hindered settling. If that were the case, the line should have a negative slope because, as the concentration of solids in suspension increased, the settling rate of the particles would undoubtedly be reduced and, consequently, the space velocity necessary to keep the solids suspended would be less. Observations in a trans-

parent pipe and results reported by Howard (4) indicate, however, another greater effect. In a pipeline carrying a suspension, the distribution of solids is not uniform over the cross-section; neither is the velocity distribution uniform, as shown in Figure 4. The normal velocity profile for turbulent flow is altered greatly by the presence of suspended solids. In one sense, the altered velocity distribution resembles that for laminar flows in that there is a considerable velocity gradient from point to point on the curve. For example, the curve in Figure 4 indicates a space velocity of about five ft. per sec. with a maximum of 6.3 ft. per sec. tapering off to about 1.5 ft. per sec. at the wall. The significant point is that the low velocities are to be found in the lower part of the pipe. Obviously, if a velocity of 4 ft. per sec. is required to effect pick-up of sand from the bottom of the pipe, the space velocity in the pipe must be much greater. It is reasonable to believe that the greater the solids concentration, the greater the distortion of the velocity profile and hence the higher is the space velocity required. This undoubtedly is the explanation for the critical velocity *versus* concentration curve shown in Figure 3.

The same argument holds for the relation between critical velocity and pipe size, shown in Figure 5.

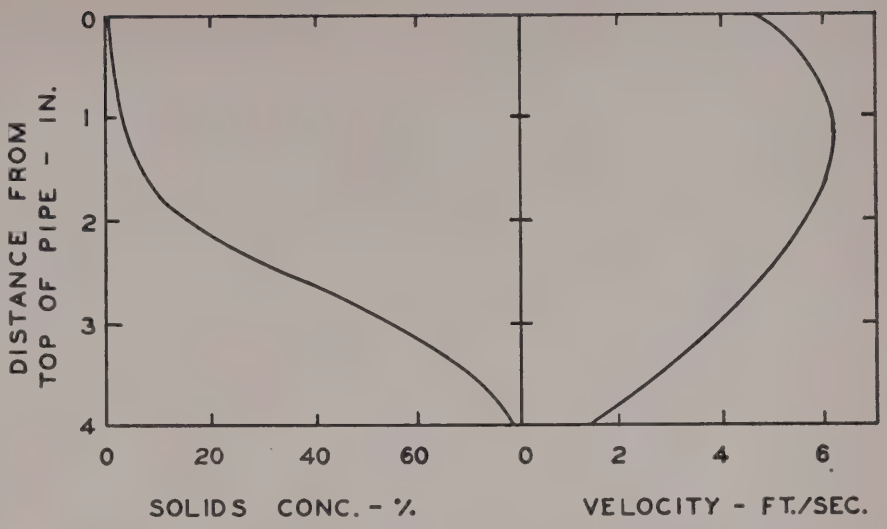

Figure 4.—Concentration gradient across a sand-suspension pipeline ***(After Howard, ref. 4).***
Data are for a 15%-solids suspension in a 4-inch pipe flowing at a mean space velocity of about 5 ft. per sec.

For a given suspension, the critical velocity increases with pipe size. The greater the solids concentration of the suspension, the greater the effect of pipe size.

The effect of temperature was found to be small but real. Figure 6 shows that an increase of 50°F. in the pipeline results in a reduction of the critical velocity, other factors being unchanged, of about 5 per cent. This would indicate that the reduction in viscosity brought about by an increase in temperature tended to restore the distorted velocity distribution profile to the normal profile.

Conclusions

The results of the present study lend considerable support to the mechanism of hydraulic transportation of solids in suspension advanced by Wilson. The suggestion that the fundamental factor in the deposition and picking-up of solids from the bottom of a pipeline is analagous to the bed-load problem of rivers has been found to be in good agree-

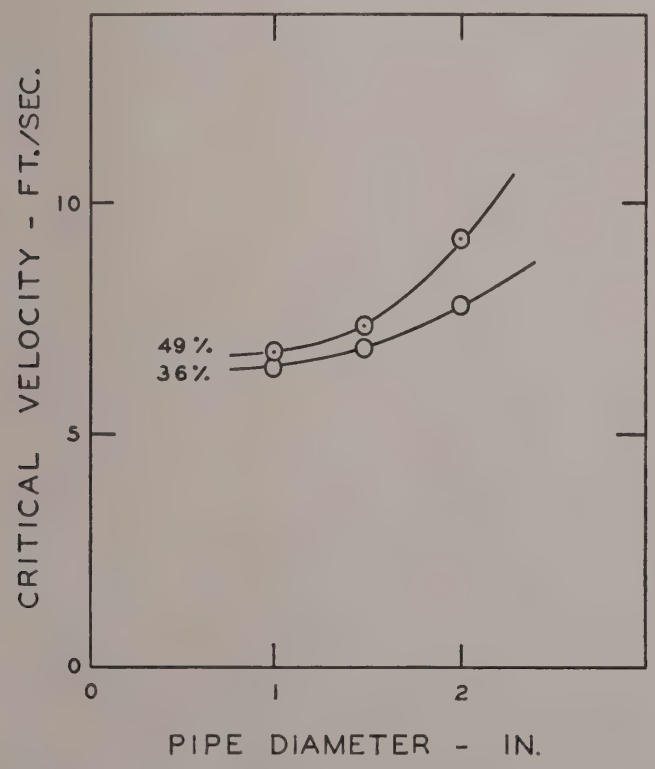

Figure 5.—Effect of pipe diameter on critical velocity of two sand suspensions, one with 49 per cent solids, the other with 36 per cent solids.

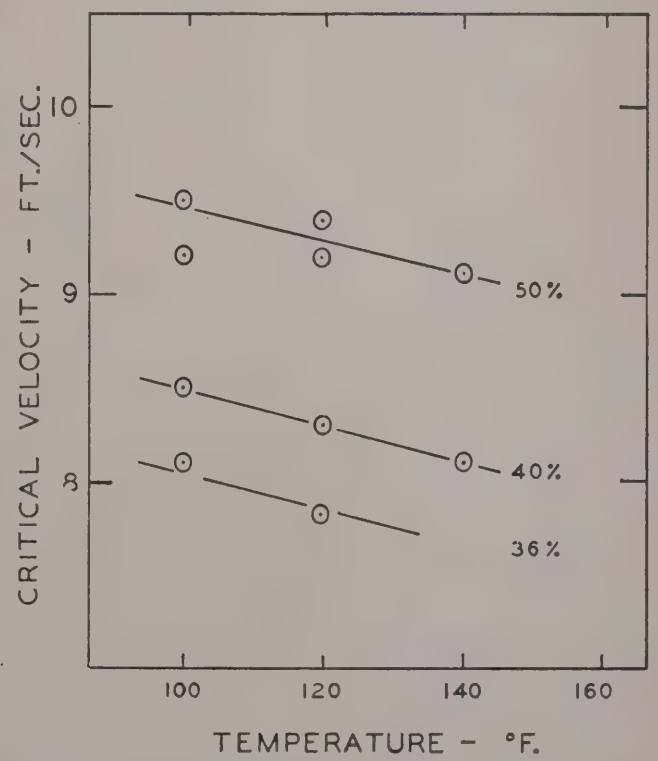

Figure 6.—Effect of temperature on critical velocity of three sand suspensions with 50, 40, and 36 per cent solids.

ment with the observations here. The 'bed-load' predictions are satisfactory in the extreme case but are subject to over-riding effects of sand concentration and pipe size, both of which tend to increase the calculated critical velocities as a result of a marked distortion of the velocity distribution profile in the pipe. The effect of temperature, though small, tends to decrease the critical velocity.

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[CORRECTED]

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